

METALLURGICAL PRACTICE

IN THE

BLACK HILLS OF SOUTH DAKOTA.

By CHARLES H. FULTON, M. E.

BULLETIN NO. 7.

OF THE

SOUTH DAKOTA SCHOOL OF MINES.

DEPARTMENT OF METALLURGY.

RAPID CITY, SOUTH DAKOTA.
JUNE, 1904.

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ERRATA

1. Page 23, foot note, third line: Read bromine for brome.
2. Page 28, line 30: Read zinc bromide for zinc bromo.
3. Page 28, line 32: Read bromo for bromide.
4. Page 28, foot note: Read Simpson for Sampson.
5. Page 23, line 12: Insert bromo before the word cyanogen.
6. Table III, test No 5, column Time of Treatment: Read "and 48 hours standing" for "48 days standing."
7. Table I, test No. 4, column Time of Treatment: Read "41 hours agitation and 31 days standing" for "60 hours agitation."
8. Table I, test 8. column Time of Treatment: Read "22½ hours agitation for "same."
9. Page 35, first table, Ore C, column "Increase in Extraction," last line: Read 6 per cent for 9 per cent.
10. Page 35, second table, in second column heading: Read "31 days contact" for "3 days contact."
11. Page 52 Table, Dakota Mill: Transfer figure "86 tons" in column "amount of solution passing while filling" to column "amount of battery solution."



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1. Sulphide Smelting at the National Smelter of the Horseshoe Mining Company, Rapid City, South Dakota.

BY CHARLES H. FULTON AND THEODOR KNUTZEN.

2. Laboratory Experiments on the Unoxidized Siliceous Ores of the Black Hills.

BY CHARLES H. FULTON.

3. The Crushing in Cyanide Solution Process as carried on in the Black Hills of South Dakota.

BY CHARLES H. FULTON.



Letter of Transmittal.

SOUTH DAKOTA SCHOOL OF MINES, {
RAPID CITY, June 5, 1904. }

SIR:—I have the honor to transmit herewith a series of papers, by Charles H. Fulton and Theodor Knutzen, on Metallurgical Processes in South Dakota.


I submit them with the recommendation that they be published as Bulletin No. 7 of the School of Mines.

Respectfully,

ROBERT L. SLAGLE, President.

Hon. Ivan W. Goodner,

President, Regents of Education.



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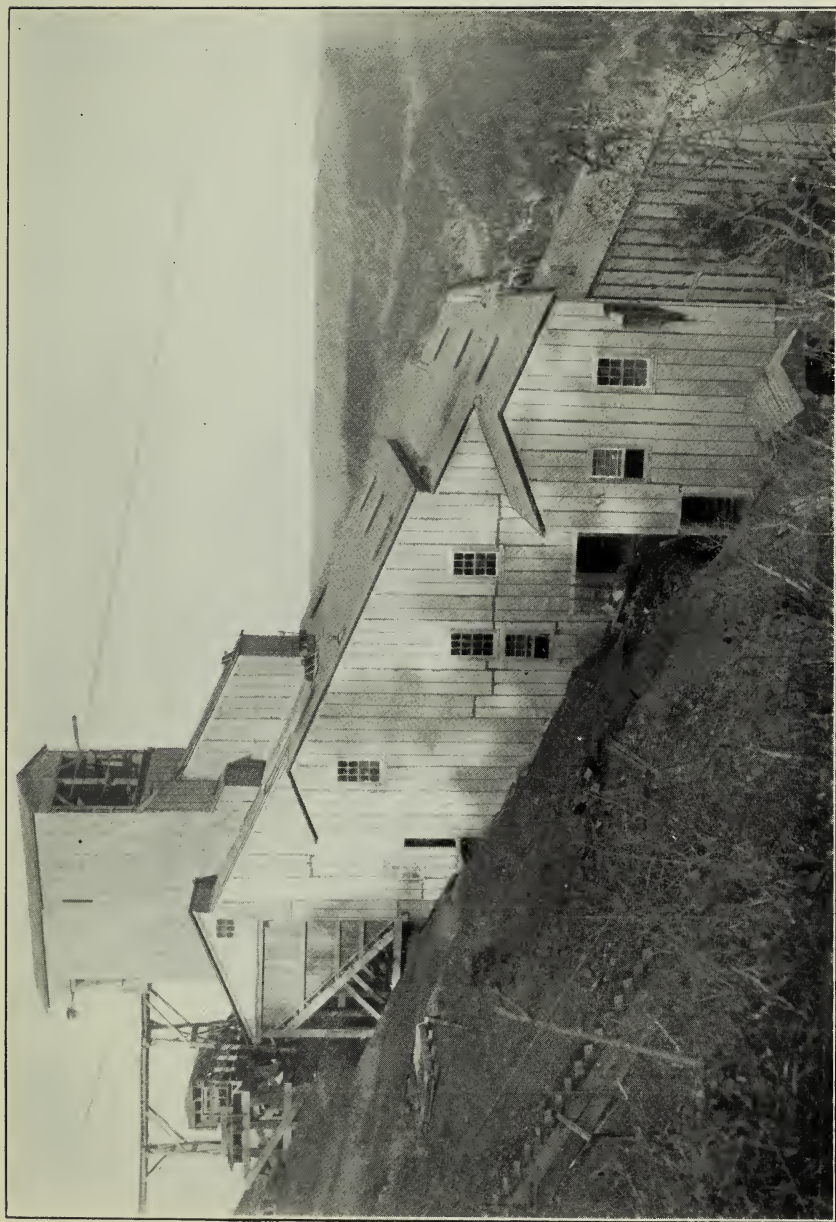
Introduction.

The successful treatment of the unoxidized siliceous ores of the Black Hills is a subject of vital interest to the mining fraternity of the region and the experiments on these ores were taken up by the Metallurgical department of the School of Mines with the idea of rendering what assistance it could in this direction to the millmen, knowing that very often their time and opportunity for this kind of work is limited.

The crushing in cyanide solution process has within the last two years assumed great importance in the Black Hills, and for that reason a detailed description of the process should prove of interest.

The writer wishes to express his obligation and thanks to all those who have so kindly and freely furnished him with information; his especial thanks being due to Mr. John Ingersol, Mr. J. V. N. Dorr, Mr. Freeman Steele, Mr. G. Howell Clevenger, Mr. John Cross, Mr. Baldwin, Mr. James Hartgering and Mr. John Millikin.

He wishes also to acknowledge the indebtedness due to Mr. William Bowman, his assistant, for the faithful and careful work done in the experimental tests on the unoxidized ores of the Black Hills, also to acknowledge his indebtedness to Mr. S. S. Arentz for much aid in the same work.



Lundborg, Dorr & Wilson Cyanide Mill, near Terry, South Dakota.

Sulphide-Smelting at the National Smelter of the Horseshoe Mining Co., Rapid City, S. D.*

BY CHARLES H. FULTON AND THEODOR KNUTZEN.

THE plant of the National Smelting Company, a corporation controlled by the Horseshoe Mining Company, was built during 1901 to smelt the dry siliceous ores of the northern Black Hills, extracting the gold and silver values in a matte of low copper-percentage which is shipped to Omaha and Denver for refining. Originally, the plant was designed to collect the values of the siliceous ores in an iron-matte, which was to be resmelted with lead-ores, in a lead furnace, the lead-bullion produced to be refined into Dore bullion in English cupelling-furnaces. However, the scarcity of lead-silver ores in the Black Hills, owing to the present non-productiveness of the Galena district, led to the abandonment of this plan; and both shaft-furnaces of the plant were run as matting-furnaces, the mattes produced being shipped as above stated.

It is not our intention to present anything very new, but rather to supplement the interesting and valuable paper, *Pyritic Smelting in the Black Hills*, by Dr. Franklin R. Carpenter.¹

Smelting in the Black Hills is a difficult matter from a commercial point of view, owing to the fact that the only productive material going into the furnace, generally, is the siliceous ore and a little copper-ore, the pyrite, or pyrrhotite, and the limestone, being barren of values, and no lime-ores, or gold or silver-bearing pyrite being at present available in the Black Hills. The recent price of coke from the east or from Colorado has also been prohibitive (\$9.50 per ton), and Cambria, Wyoming, coke (\$4.50 per ton) is of such inferior quality that it cannot be used alone, but has to be mixed in the proportion of 2 to 1 with

¹ *Trans.*, xxx., 764.

* [Reprinted from the Transactions of American Institute of Mining Engineers February, 1904.]

eastern or Colorado coke, to be able to smelt with it at all.

The National Smelting Plant is situated at the eastern end of Rapid City, on a terrace site on a spur of the Fremont, Elkhorn & Missouri Valley Railroad, a branch of the Chicago & Northwestern Railway system. Directly below the railroad-trestle are nine 125-ton bins: Three for siliceous ore, 2 for limestone, 2 for coke, and 1 for coal. The bottom of the bin slopes 50° , the planking being protected from wear by railroad-iron, placed transversely every foot.

The material is shipped to the smelter in 20-ton ore-cars, usually having a bottom-discharge. It is sampled by shoveling on a sampling-floor at the top of the bins. Lots of 60 tons or less are sampled by taking every fifteenth shovel, while those in excess of 60 tons are sampled by taking every twentieth shovel. The sample is thrown down a chute at the side of each bin, carried by a barrow to the sampling-works, and is there crushed in a 9-by 15-in. Blake crusher which divides it into halves by an "A" discharge, one-half going directly to a pair of 24- by 12-in. Allis-Chalmers rolls. The discharge from these rolls is reshoveled, every fifth shovel being taken as the sample, or every tenth shovel, if the ore lot is more than 60 tons. The sample obtained in this way is crushed in a pair of 12- by 12-in. sample-rolls, then coned and quartered on a plate-floor, and the resultant sample ground in a sample-grinder. The moisture is determined at once on the sample-floor, in a specially provided drying-cup-board.

FURNACES—There are two blast furnaces, one, a copper-matting furnace, 144 by 38 in. in cross-section at the tuyeres, and 15 ft. in height to the downtake, and the other, a lead-furnace, 120 by 36 in. in cross-section at the tuyeres, and of the same height as the copper-matting furnace. The lead-furnace is used as a matting-furnace by bricking up the well. Both furnaces have removable hearths placed on trucks, and the off-take at one end just below the feed-floor. They are charged by a specially designed bottom-dump charge-car, running directly over the furnace-top. In our opinion this method of charging is the most desirable one. No trouble from fumes is experienced on the charge-floor. The furnaces, however, could easily be increased 4 ft. in height, which would enable better work to be done in them.

Steam is furnished by two 200-h. p. Stirling water-tube

boilers, with a steam pressure of 125 lb. per square inch. A No. 8 Green blower furnishes blast for the large furnace, and a No. 7 blower of the same type for the small furnace. Both blowers are directly connected to horizontal engines on the same bed-plate. The blowers run from 130 to 150 rev. per min., furnishing blast at a pressure of from 14 to 18 oz. per square inch. The blast main from the large blower is 30 in. in diameter, and that from the small one 24 inches.

The coal is of a very poor grade, being slack from Cambria, Wyo., which costs \$2.75 per ton, delivered at the plant. The ashes are sluiced from the boiler-plant through launders to the slag-dump. The water-supply of the plant is obtained by pumping from Rapid Creek, a few hundred feet below the plant.

SLAGS—Table I, gives the analyses of typical slags made at the plant.

TABLE I—*Analyses of Slags.*

Kind.	Silica.	Ferrous Oxid.	Lime.	Alumina	Zinc Oxide.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Blowing in slag.....	42.65	16.67	30.7	6.85
Typical slag.....	47.5	18.7	28.25	3.5
Typical slag.....	50.2	16.35	28.30	4.2
High alumina and zinc.....	42.86	17.24	26.61	9.39	2.28

The precious metal-content of the slags given in Table I are usually, gold 0.01 oz., and silver 0.20 oz. per ton, a trace of copper also being present.

The limestone used as a flux is very pure, as shown by the analyses given in Table II, and contains only a trace of magnesia. Some magnesia, up to 8 or 10 per cent replacing lime, would be desirable, owing to its greater silica-saturating power, and the lesser specific gravity of the resultant slag. The Golden Reward plant at Deadwood uses a magnesian limestone successfully. At the present time no limestone containing magnesia is available in the vicinity of Rapid City.

The slags are fluid and flow readily from the furnace with a slight arch. They chill quickly, which indicates a rather high temperature of formation. Water-cooled slag-spouts have been tried on the furnaces, but had to be discarded owing to their marked chilling-effect on the slag. The greater part of the slag is granulated by waste water from the furnace-jackets and discharged to flat-cars of the railway which utilizes it as road-

ballast. The slag flows from the settling-pot in a thin stream, falling from a height of 4 ft. and strikes the water which flows in a heavy cast-iron gutter of semi-ellipsoid section, 8 in. wide and 6 in. deep, inclined 3 in. per foot for the first 10 ft. The section of the gutter beyond the first 10 ft. is of a larger cross-section, and inclined but 1 in. per foot. Owing to excessive wear, the section of the gutter where the slag strikes it has to be frequently renewed.

Heated blast is used in smelting. The blast heating apparatus is a U-pipe stove, containing 12 U-pipes each 16 in. in diameter and 10 ft. high. This type of stove is not as efficient as it might be owing to the difficulty in preventing leakage. The stove is placed in the dust-chamber directly beneath the down-take, it being intended to heat the blast only by the waste heat from the furnaces. Under ordinary conditions the temperature of the blast, taken at the tuyeres, is 131° Fahr., with the outside air at 65° Fahr. With the cupelling-furnaces, running on some experimental work, the temperature of the blast at the tuyeres was as high as 320° Fahr., with the outside air at 77° Fahr.

The flue of the plant is of the zig-zag type, 350 ft. long, extending up hill to a plate iron stack, 10 ft. in diameter at the top, and 166 ft. in height from the bottom of the flue where it merges into the stack. The total height from the tuyere-level to the top of the stack is 275 feet.

TABLE II—*Composition of Materials of Furnace-Charge.*

Name of Material.	Silica.	Alumina.	Ferrous Oxide.	Iron.	lime.	Magnesia.	Sulphur.	Carbon.	Copper.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Montezuma pyrite	19.35	7.63	31.7	2.57	0.30	32.3	6.50
Bion pyrite	23.95	31.75	37.03
Penobscot ore	61.44	29.2 (a)	5.9	trace	3.01
Ben Hur ore	62.56	14.7 (a)	14.54
Ben Hur ore	73.11	12.36	7.13 (1)	7.2
Limestone	1.94	0.68 (a)	5.61	0.63
Montana copper-ore.....	31.62	21.47	22.85	24.16
Ashes:									
Cambria coke, 30 per cent....	36.57	40.35	14.1	4.01
Fairmount coke	60.46	37.0	2.75	trace
Ashes, 12.5 per cent.

(a) Includes ferric oxide.

(1) Ferric oxide.

The Penobscot ore contains from 0.88 to 0.96 oz. gold and from 1 to 2 oz. silver per ton. The Ben Hur ore contains from 0.73 to 0.80 oz. gold and 1.5 oz. silver per ton. It is generally aimed to keep the ore-value not less than \$20 per ton.

MATTE—The first matte produced is low in copper, containing from 10 to 14 per cent copper and from 4 to 5 oz. gold and from 6 to 7 oz. of silver per ton. The matte-fall figured on the total furnace-charge is from 4 to 5 per cent, this fall being amply sufficient to collect the values. We believe a 3 per cent matte-fall would be sufficient. It was found by experience that some copper was absolutely essential in order to have the matte collect all of the values and at the same time to produce sufficiently clean slags. In 1891, shortly after the plant was started, it was endeavored to run without copper-ores, owing to the difficulty of procuring them. Mattes were made with only a trace of copper in them, but the slags invariably contained from \$1.00 to \$2.50 in value per ton in gold. This loss being too great for profit, copper-ores had to be procured. Upon the addition of copper-ores to the furnace, the abnormal loss of value in the slag disappeared, dropping to the normal value of from 20 to 30 cents per ton and often less. It has been demonstrated by Dr. R. Pearce² and E. G. Spillsbury³ that iron sulphide will not collect gold and silver. Metallic iron will collect gold, but practically no silver, as Dr. F. R. Carpenter has pointed out⁴—a fact which is amply proven by the sows, or metallic-iron accretions, formed in the hearth of the furnace as well as in the fore-hearth. The matte formed rarely contains more than 30 per cent sulphur, while the iron monosulphide contains 36.36 per cent, so that the matte is evidently a subsulphide. It also contains metallic iron, which can readily be abstracted by the magnet. We agree with Dr. Carpenter that it is this metallic iron in the matte which collects the gold, but, unfortunately, it is rarely in the matte in sufficient quantity to give clean slags. Paradoxical as it may seem, the quantity of metallic iron formed in the furnace is due to a large extent to the amount of oxidation which takes place in the furnace. This point is referred to later in this paper under the section devoted to sows.

The amount of oxidation being difficult of control, the matte-composition and matte-fall vary from time to time. There are occasions when practically no matte is being made, but at the same time the slags do not increase in value, showing that while no matte is made, metallic iron is being produced. This condition of affairs occurs during periods of much oxidation, and is usually remedied by the charging of extra quantities of pyrite,

² *Trans.*, xviii., 454

³ *Trans.*, xv., 767.

⁴ *Ob. cit.*

in order to furnish more sulphur, and leave some to remain in the matte.

Accretions on the furnace-walls will decrease the quantity of matte made, by raising the zones of oxidation. Of course, the quantity and pressure of the blast, also, greatly influence the matte production.

DESULPHURIZATION—The matte-fall being generally but 4 or 5 per cent. shows the great desulphurizing action of the furnace, which amounts ordinarily to from 70 to 77 per cent. To take the workings of a typical day, the quantity of sulphur fed into the furnace in the shape of pyrite was 7,350 lb. and the sulphur in the matte was 2,100 lb., showing a loss of sulphur of 5,250 lb., which is equivalent to 71.5 per cent of the total quantity in the materials charged into the furnace.

The desulphurization-figures for December, 1903, were as follows:—The quantity of sulphur fed into the furnace in pyrite, 175 tons; in matte, 37.5 tons; in copper-ore and concentrates, 27 tons, a total of 239.5 tons. The sulphur in the matte produced was 52 tons, showing a loss of sulphur of 187.5 tons, or 78 per cent of the quantity charged into the furnace.

The sulphur in the materials carried over mechanically in the flue-dust may be disregarded on account of its relatively small quantity and the fact that much of it is in an oxidized condition. The total quantity of flue-dust produced in treating 3,952 tons of charge amounted to 10 per cent, or 395.2 tons, having an average sulphur-content of 3 per cent, which is equivalent to 11.8 tons of sulphur, a large portion being in an oxidized form.

The matte produced without copper-ore was made with the furnace running on pyritous material of the following composition. Iron, 24.52; silica, 27; lime, 3.06; lead, 5.82; zinc, 8.55; sulphur, 28.03; arsenic, 3.4 per cent; copper, trace; gold, 0.04 oz., and silver, 2.86 oz. per ton.

TABLE III.—*Composition of Mattes.*

Kind.	Iron.	Copper.	Sulphur.	Zinc.	Gold.	Silver.
	Per Cent	Per Cent.	Per Cent.	Per Ct.	Ounces Per Ton	Ounces Per Ton
Made without copper ore.....	61.5	1.6	2.18	7.1	9.7
Matte made with little copper-ore.....	68.5	3.5 (a)	27.9	4.2	8.7
Matte.....	5.57	17.65	10.25
Typical Matte.....	22.6	30.0	17.11	21.4
Typical Matte.....	20.8	18.73	19.7
Typical Matte.....	19.3	11.62	18.76

(a) The slag accompanying this matte had an assay value of \$1.40 per ton.

It is worthy of note that while some of the zinc enters the matte, practically no lead does. The analysis of the slag corresponding to this matte is given in the Table I under the name, "High Alumina and zinc." The quantity of copper in this matte, 1.6 per cent, is insufficient to give a clean slag, which in this instance had a value in gold of \$1.20 per ton. When operations were first started at the plant an experiment was made of adding lead-ores in quantities equaling those of copper-ores now added, to ascertain whether lead would enter the matte and reduce the abnormal losses in the slag. However, no lead was found in the matte, for the reason that the conditions under which the furnace was operated precluded its entrance into the matte. There is so much oxidation that most of the lead becomes volatilized to the great detriment of the yield of silver. Lead, even in small quantities, is very undesirable in sulphide smelting.

The first matte produced is generally resmelted twice, the third matte being the shipping-matte. The Table IV shows the concentration:

TABLE IV.—*Concentration of Gold and Silver in Matte.*

Matte.	Copper.	Gold.		Silver.	
	Per Cent.	Oz.	Per Ton.	Oz.	Per Ton.
First.....	13.5	4.04		6.03	
Second.....	21.5	10.05		15.6	
Third.....	22.6	17.11		21.4	

The matte is cast into slabs in cast-iron molds, in order to break it up readily and have it in a convenient form for shipping.

AMOUNT OF COPPER NECESSARY TO MAKE CLEAN SLAGS—At the present time, copper-ore for the matte is brought from Montana at a considerable expense, and is added in just sufficient quantity to produce the desired effect. It is aimed to have at least 10 lb of copper in the charge for every ounce of gold present, and more, if a supply is available.

The silver is much more affected by the lack of a certain proportion of copper than is the gold. The ratio of silver to gold in the ore is from 1.5 to 2, to 1, and, with an equal saving of both metals, this ratio should be preserved in the matte. Mattes containing less than 20 per cent of copper show a distinct loss of silver. In general, we do not think that sulphide-smelting is adapted to a close saving of the silver.

FUEL—Owing to the high melting-point of the siliceous slags, the quantity of carbonaceous fuel is considerable. The

fuel expressed in percentage of total charge seems very high, but is explained by the very poor quality of the Cambria coke used, the ash amounting to about 30 per cent. Generally a mixture of two-thirds of Cambria coke and one-third of Eastern or Colorado coke is used. The quantity of coke used varies from 14 to 18 per cent of the ores and flux charged into the furnace.

TABLE V.—*Composition of the Coke Used at the National Smelter*

Kind of Coke.	Fixed Carbon	Volatil'e Carbon	Ash. (a)	Water.
	Per Cent.	Per Cent	Per Cent.	Per Cent.
West Virginia.....	83.83	10	12.42	0.69
Cambria, Wyo.....	65.22	3.93	29.93	0.92
Colorado.....	86.86	1.7	10.7	1.6

(a) The analysis of the ash is given in Table II.

The statistics given in Table VI show a capacity of the small furnace of about 105 tons of burden per day. The large furnace has a capacity of about 130 tons of burden per day. Both furnaces operate under the disadvantageous condition of treating a very large quantity of fines, aside from using such very poor and friable coke.

SOWS—In our opinion the production of sows is practically inseparable from sulphide-smelting when high concentration is done. The sows are due to the strong oxidizing effect of the

TABLE VI.—*Capacity of the Furnaces of the National Smelter.*

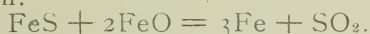
Material.	Smaller Furnace. November, 1903.	Smaller Furnace. December, 1903.
	Tons.	Tons.
Siliceous ore.....	1,100	980
Pyrite.....	512	548
Limestone.....	1,038	1,070
Copper ore etc.....	70	90
Flue-dust (a).....	250	348
Matte.....	136	125
Total burden.....	3,126	3,161
Coke.		
Cambria.....	510	517
Eastern.....	180	162
Fuel percentage.....	18	17

(a) Including accumulated flue-dust from the chlorination works.

furnace, as shown by the following data. A desulphurization of 80 per cent; the production of copper sulphate, found in layers in the accretions of the downtake; no evidence of carbon mon-

oxide in the furnace gases; the volatilization of all the lead fed into the furnace, and the facts that no iron goes into the furnace as oxide, and the slag contains from 18 to 20 per cent of iron oxide in the form of silicate. These data make it difficult to imagine that the reducing conditions in the furnace could exist sufficiently strong to produce metallic iron.

We believe that the sows are produced by oxidation in a similar way that metallic copper is produced during bessemerizing, taking as the first stage, the melting of the pyrite, FeS_2 ; and the loss of the one atom of sulphur forming the monosulphide FeS ; the second stage, the gradual oxidation of the sulphur in the monosulphide, producing a subsulphide; the third stage, the production of some ferrous oxide, part entering the slag, and part reacting with the sulphide present, producing sulphurous acid gas and metallic iron, according to the following chemical equation:



Experience has shown that a larger quantity and higher pressure of blast result in an increased production of metallic-iron sow, and, from its analysis, it is seen that it contains practically no carbon, which apparently should be present if the metallic iron was due to the reducing action of the coke.

In the large furnace (38 by 144 in. in cross-section at the tuyeres), a 15-ton sow was produced in a 7-months' run, having an approximate value of \$5,000 in gold. In the smaller furnace in a 3-months' run, under a lower blast-pressure, a 5-ton sow was produced, having an approximate value of \$1,500.

The sow, as a whole, is not homogeneous, and consists mainly of metallic iron containing intermixed slag and a little matte. The metallic iron contains practically no silver, but considerable gold. The iron has a crystalline structure similar to that of pig-iron, possesses a distinct silver color and is practically pure metallic iron. In the large sow some pieces of copper were found. Rarely a small button of lead is found in a sow.

TABLE VII.—*Composition of Sow Produced at the National Smelter.*

Material.	♦ Iron.	Sulphur.	Gold.	Silver.	Copper.
	Per Cent	Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Per Cent.
Average Sow			20.3	2.4
Crystalline iron	99.68	0.2	29.6	4.4	trace
Pore-hearth sow (a)			35.8	nil	14.13
Lead in sow			134.17	273.45
Copper in sow			83.8	nil

(a) A true sow.

Material from the fore-hearth resembling a sow contained, iron, 72.3; sulphur, 19.19; copper, 6.94 per cent; gold, 6.3 oz., and silver, 5.2 oz. per ton. It is usually through the accumulation of this material and that of the true sow that the fore-hearth is lost. The top layer of the sow, as well as that part in contact with the fire brick, is usually of an oxidized appearance.

The treatment of the sows for the recovery of the gold and silver values in them is a difficult problem, especially when no reverberatory furnaces are available. The National Smelter has not this valuable adjunct which is practically necessary for the treatment of the sows and the flue-dust. At the present time the sows are broken up by blasting, a very expensive operation, and re fed into the furnace a little at a time with the pyrite in order to resulphurize the iron.

FLUE-DUST—Owing to the absence of reverberatory furnaces, the flue-dust made amounting in quantity to about 10 per cent. of the charge is resmelted in the blast-furnace, a rather undesirable procedure. Aside from this flue-dust, the plant has treated at times considerable quantities of concentrates from stamp-mills, and accumulated flue-dust from chlorination-plants, so that the quantity of fines was really more than the furnace could profitably handle. In order to put this material through and keep the quantity of flue dust produced within limits, the furnace low in itself, was operated with a low charge, and consequently with a fairly hot top, which accentuated the losses in fume. The flue-dust increases in value as the stack is approached.

TABLE VIII.—*Analysis of Flue-Dust.*

Material.	Fe ₂ O ₃ .	SiO ₂ .	Lime.	Sulphur.	Copper.	Gold.	Silver.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Oz. Per Ton.	Oz. Per Ton.
From dust-chamber.....					0.88	1.02	4.1
Beginning of flue (a).....	35.14	35.7	4.12	5.60	0.80	1.17	12.60
Average flue-dust (b).....	25.13	34		4.63(c)	0.80	1.25	13.55

(a) Contains also Al₂O₃ 4.07 per cent.

(b) Contains also 9 per cent carbon

(c) Considerable of which is soluble in water.

TABLE IX.—*Analysis of Accretions in the Dust-Chamber.*

Place.	Carbon.	Copper.	Gold.	Silver.
		Per Cent.	Oz. Per Ton.	Oz. Per Ton.
From the bottom of downtake.....	Considerable	14.1	0.70	8.80
On the blast-heating apparatus.....	Considerable.	16.1(a)	0.40	5.1

(a) All soluble in water, for the most part being present in the form of copper sulphate.

The ratio of gold to silver in the ore is usually about 1 to 2, so that the analysis of the accretions shows a heavy loss of silver by volatilization. In fact, the process as a whole is unfavorable to a high recovery of the silver, especially if high concentration is carried on.

LOSSES IN FUME—It has been the experience at the National Smelter that there is a considerable loss of values by smoke and fume, especially in silver, and gold, under certain conditions; as, for instance, when operating with high concentration and lead or zinc in the charge. Lead is not at all desirable in the furnace, most of it being volatilized, carrying values with it.

TABLE X.—*Analysis of Condensed Fume from the National Smelter.*

Place from which Taken.	Gold.	Silver.	Copper.	Remarks
	Oz. Per Ton	Oz. Per Ton	Per Cent.	
Flue near the sack	1.50	15.50	Trace
Middle of flue	1.60	11.90
Near the beginning of the flue	1.60	10.40	Contains lead, soluble sulphates, arsenic, some calcium sulphate.
Average value of fume from the steel roof of the flue	2.00	16.1	

The analyses show the relatively much greater volatilization with the silver suffers, although it is evident that the gold in the ore also suffers loss. As a matter of experience, it might be said that when the furnace is running low with a hot top, and some of the above mentioned undesirable elements present, the monthly account based on the ore-assays, will show a considerable loss in silver, and a very appreciable one in gold, as well as some copper, which are not slag losses. Greater attention paid to the saving of fume in sulphide-smelting plants would lead to economy.

Laboratory Experiments on the Unoxidized Siliceous Ores of the Black Hills.

BY CHARLES H. FULTON.

It is well known that while the "red" or oxidized siliceous, or Potsdam ores of the Black Hills yield readily from 75 to 85 per cent of their gold value to the cyanide process as at present practiced in the Hills, the "blue" or unoxidized ores yield but from 30 to 50 per cent extraction, and are as a general thing the bugbear of the millman. Regarding the occurrence of the "blue ores" in the ore bodies, no distinct line can be drawn separating them from the red ores, the blue ores often being found in bunches and pockets in the red ores, in widely varying quantities. The amount of blue being mined is, however, increasing as the workings penetrate further into the ore bodies. This fact is amply proven by the extraction figures of the various mills which, as a general thing, have dropped from 5 to 10 per cent. within the last two years.

While, as a usual thing, the siliceous ores present the same general characteristics at all of the properties yet ores from the individual mines will vary quite widely as regards amenability to cyanide treatment. The ores can be roughly divided into two general classes, the shale ores and the quartzite ores, of which usually, though not invariably, the quartzite ores are the more refractory, generally on account of their greater compactness and density. There are some blue ores that will not yield more than 20 to 25 per cent of their values to ordinary cyanide treatment, while others will yield 50 to 60 per cent, although the latter percentage of extraction is rare.

The gold and silver values are very evenly distributed throughout the ores as is shown by the assay of different screen sizes*. It is rare that the dust has a higher value than the coarser sizes, and it sometimes has a lesser value. As an example, the dust accumulated on the mill rafters from the dry crushing of ore at the Imperial mill, had a value considerably

*F. C. Smith, *The Occurrence and Behavior of Tellurium in Gold Ores*, T. A. I. M. E. Vol. 26, p. 491. Similar Experiments made in the laboratory of the School of Mines more recently confirm these results.

less than that of the ore crushed during the time of its accumulation. These facts point to the occurrence of the gold in the ores in a comparatively non-brittle mineral. Just in what form the values do occur has been much speculated on† but it cannot be said that any definite conclusions have been reached. It seems a rather important point to determine, from a metallurgical point of view, as having an influence on the mode of treatment. F. C. Smith states that he has found tellurium in many of the Potsdam ores, and believes that the gold occurs in many instances as sylvanite, although this mineral has never been isolated except in ores from the Ironsides mine, Squaw creek.† However, the fact that many analyses made at the School of Mines recently, fail to disclose tellurium or but traces of it, in many ores, lead to believe that tellurium is not of such widespread occurrence in the siliceous ores, as believed, and that the gold bearing mineral is not in most cases a telluride. Experience at Cripple Creek, Colorado, and at Kalgoorlie, Australia, shows that the dust from telluride ores invariably runs higher in value than the original ore. As an example, a Cripple Creek telluride ore shows this result: ‡

Original ore value = 0.82 oz.

Dust, † 200 mesh. value = 1.04 oz.

Dust, - 200 mesh. value = 1.92 oz.

This is due to the extreme sectility of the telluride minerals. As already stated the dust from the Black Hills[§] siliceous ore rarely has a higher value than the sands.

It has also been found that a cyanide solution plus bromo cyanogen exerts a solvent action on calaverite and possibly similar telluride minerals, as shown by results on a large scale at Kalgoorlie, and by laboratory tests on Cripple Creek telluride ores. For example: A Cripple Creek ore (Vindicator mine) having a value of 1.09 oz. gold, gave 40.4 per cent extraction on raw ore by a 0.4 per cent cyanide solution. A 0.4 per cent cyanide solution plus bromo cyanogen, same time of treatment, gave 91.7 per cent.*

† H. M. Chance, The Discovery of New Gold Fields, T. A. I. M. E. Vol. 29, p. 229, 1933, 1937.

† F. C. Smith, T. A. I. M. E. Vol. 29, p. 1933.

‡ Figures furnished by Mr. John Millikin, Deadwood. The dust mentioned is from the frame work of a Cyanide plant at Florence, Colorado.

* For data concerning Kalgoorlie ores, see "The Diehl Process" by H. Knutzen, T. I. M. & M. June, 1902.

An extended series of experiments with bromo cyanogen (results of which follow) on the "blue" siliceous ores of the Black Hills fail to show such phenomenal results, although an increase of extraction of from 8 to 10 per cent is noticeable.

These facts certainly throw doubt on the supposition that the gold in the siliceous ores is generally in the form of a telluride.

Below are appended a number of analyses of siliceous ores, in which particular attention has been paid to the elementary constituents present in small quantities such as arsenic, antimony, etc.

No. 1 † (blue ore.)		No. 2. † (blue ore.)	
Gold.....	0.63 oz. per ton	0.85	oz per ton
Silver.....	2.00 " " "	6.08	" " "
Silica....	65.38 per cent	80.00	per cent
Iron.....	13.40 " "	7.50	" "
Sulphur....	11.40 " "	4.40	" "
Arsenic....	0.90.....	2.00	" "
Antimony...	trace	0.00	
Tellurium...	0.003.....	trace	
Zinc.....	0.00.....	0.00	
Copper.....	0.02.....	0.004	
Manganese...	trace.....	0.54	
Alumina..	5.43.....	1.79	
Lime.....	2.10.....	1.70	
Magnesia..	0.20.....	not determined	
No. 3. † (blue ore.)		No. 4. † (partially oxidized) ‡	
Gold.....	3.35.....	2.00	
Silver.....	1.75.....	0.62	
Silica....	80.90.....	84.80	
Iron.....	9.94.....	7.50	
Sulphur....	4.53.....	0.75	
Arsenic....	0.29.....	0.00	
Antimony...	trace.....	0.00	
Tellurium...	0.007.....	trace	
Copper.....	0.013.....	0.008	
Zinc.....	trace.....	0.00	
Manganese..	trace.....	0.96	
Alumina....	1.70.....	1.02	
Lime.....	0.50 per cent	0.90	per cent
Magnesia...	trace.....	not determined	
Ore "A" No. 5. (blue ore.)		Ore "B" No. 6. (blue ore.)	
Gold....	0.78 oz. per ton	0.90	oz. per ton

‡Kindly furnished by Mr. John Gross, Maitland, S. Dak.

Silver.....	—	—
Silica.....	77.38	93.72 per cent
Iron.....	3.54	2.67 “ “
Sulphur.....	4.42	0.69 “ “
Arsenic.....	0.55	0.002
Antimony....	trace	0.0893
Tellurium....	none	none
Copper.....	trace	none
Manganese....	none	0.082
Alumina.....	2.80	3.53 per cent
Lime.....	0.56	—
Magnesia....	trace	—
Phos. acid....	0.32	0.0059
Soda.....	1.32	—
Lead.....	—	trace
Thallium.....	?	?
Tungsten....	—	none

In the analyses, copper, antimony, arsenic, tellurium, will be noticed in small quantities in most of the ores. In the Maitland ores bismuth also exists in appreciable quantities for at the Penobscot mill it is found in considerable amounts in the zinc precipitates. Certain peculiar facts are noticed in roasting various blue ores, as a general thing a fair extraction may be obtained on *most* of the blue ores if they be finely crushed and roasted at a low heat for a considerable time. If the heat be raised to a high temperature at once, with some ores but little better extraction can be obtained than from the raw ore. Other blue ores again do not behave in this way. It seems possible that the gold and silver are held in some complex mineral combination into which arsenic and antimony enter, so that a high temperature and oxidizing conditions transform this compound into stable compounds, failing to liberate the gold and silver. The gold and silver bearing mineral is also very uniformly distributed throughout the rather dense ores, so that in most instances a comparatively fine crushing* is required to liberate the values from the rock mass. Roasting of the ores causes practically no loss of values.

The fact that the siliceous ores of the Black Hills were possible telluride ores led to the rather extended series of experiments with bromo cyanogen, which while not giving the results hoped for, aid the extraction of the values somewhat, so that the publishing of the results seems justified. All the tests made were agitation bottle tests, which, is the usual starting point in

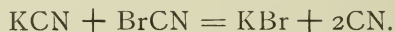
*There are exceptions to this rule, as for instance, certain ores of the Wasp No. 2 mine, and the Ragged Top ores,

experiments of this kind. The results obtained by these were to be confirmed by tests on a larger scale, but unfortunately the time, means and apparatus for this work was not available. The tests were made practically all on the "blue ores" with the idea that the more oxidized ores would be tested later. The effect of bromo cyanogen on the extraction is soon to be tried on a large scale by one or two of the mills of the district, and the results will be watched with great interest.

BROMO CYANOGEN—Bromo cyanogen and chloro-cyanogen were first used unsuccessfully in the Cyanide process by Dr. William H. Gaze, in Australia in 1892. Later in 1894 the use of bromo cyanogen was patented by Messrs. Sulman and Teed in connection with zinc dust precipitation in England. It is at present used in Kalgoorlie, West Australia, in the Diehl process with success, on sulpho telluride ores.†

Bromo Cyanogen above a certain temperature is a pungent irritating colorless vapor, extremely poisonous which affects the eyes and lungs in a very irritating manner. At 60 °C it sublimes into colorless needles, which afterwards change to cubes. The vapor and sublimate is soluble in water and methyl alcohol, more so in alcohol than in water.

When heated in a closed tube to 140 °C it is converted into $\text{CN}_3 \text{Br}_3$. It is decomposed by potassium cyanide with the following reaction.



With alkalis such as potassium hydrate, the following reaction probably takes place in dilute solution.



PREPARATION OF BROMO CYANOGEN—There are three methods of preparing bromo cyanogen.

1. *By the Addition of Dilute Bromine Water to a Dilute Solution of Potassium Cyanide, the Last Being in Excess*—The amounts of bromine and cyanide are best present in the theoretical quantities expressed by the reaction.

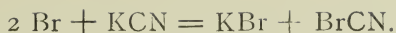
†The Diehl Process, H. Knutzen, T. I. M. & M. June 1902.

Metallurgical Progress in W. Australia, E. & M. Jour. Vol. 75, p. 18,251, also Vol. 77, p. 31.

The Treatment of Sulpho-telluride ores at Kalgoorlie, E. & M. Jour. Vol. 76, p. 156.

The Treatment of Telluride ores by Dry Crushing and Roasting at Kalgoorlie, T. I. M. & M. Oct. 15, 1903.

Cyanide Practice, by Alfred James, 1903.



The bromine water may be added until a faint permanent yellow color appears. The cyanide solution during the addition of bromine water should be kept cold by ice water, as the heat developed by the reaction is sufficient to drive off the bromo cyanogen as a vapor from the solution.

2. *Preparation by Means of Bromine and Mercuric Cyanide.*—When one part of liquid bromine is allowed to flow gradually on 2 parts of mercuric cyanide (dry salt) in a retort* surrounded by ice, bromo cyanogen and mercuric bromide are formed with a great evolution of heat. Bromo cyanogen sublimes in needles contaminated with free bromine, which, however, flows back into the retort and enters into complete combination. Gentle heat, by means of an alcohol lamp, is then applied and the BrCN sublimed into a receiver surrounded by ice water.

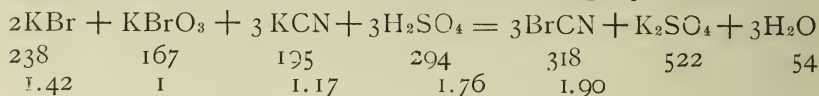
Mercuric cyanide, if not available, can be made by dissolving mercuric oxide, (which can be made from mercury or mercuric nitrate) in a solution of hydrocyanic acid. The hydrocyanic acid can be made with potassium Ferro cyanide or potassium cyanide and dilute sulphuric acid, by heating in a distilling bottle and collecting the vapor in ice water, or water cooled by ice. When the mercuric oxide is all dissolved in the hydrocyanic acid solution in the cold it is gently evaporated to crystals on a water bath, care being taken not to remove the very last of the solution on the bath, but to let it evaporate in the air, or in an air bath at a temperature not exceeding 120° Fahr., otherwise there is danger of reducing the mercuric cyanide.

Bromo cyanogen crystals obtained in this way can be kept in a tightly corked bottle in a cold place for use indefinitely. This method is the most difficult and expensive method of preparation, but yields the pure crystals of bromo cyanogen unadulterated by any other salt. This may sometimes be desirable for certain tests, but for general experimental purposes bromo cyanogen made by the first method, or still better by the third method, to be described, which is the commercial method of its production in Western Australia, is just as desirable.

3. *Preparation by Means of Bromine Salts, Potassium Cyanide and Sulphuric Acid.*—When solutions of potassium

*The apparatus for this work is best a small retort and receiver, the retort having an opening for a small thistle tube with stop cork for the introduction of bromine. See Denver Fire Clay catalogue, p. 173, No. 1313.

bromide, potassium bromate, potassium cyanide and sulphuric acid are mixed in the proper proportions, bromo cyanogen in solution is produced according to the following equation:



It is essential for success that the proportions of the theoretical equation be followed closely and that the solution be not too concentrated.

TO TAKE A DEFINITE EXAMPLE:

Taking as the starting point 25 grms. of potassium bromate it will require

$$\begin{aligned} 1.42 \times 25 &= 35.5 \text{ grms. of potassium bromide} \\ 1.17 \times 25 &= 29.25 \text{ " " " cyanide} \\ 1.76 \times 25 &= 44.0 \text{ " " " sulphuric acid.} \end{aligned}$$

This should produce 47.5 grms. of bromo cyanogen. The Potassium bromide and bromate may be dissolved in about 400 c. c. of cold water. It will take some time to dissolve the bromate, as it is rather difficultly soluble in water. The potassium cyanide can be dissolved in 200 c. c. of water, and the required amount of sulphuric acid diluted to 400 c. c.

The strength of the sulphuric acid should be estimated before dilution with an N-10 solution of sodium hydrate, 4 grams to the litre, one c. c. of which is equivalent to .0049 grams of sulphuric acid. The proper amount of sulphuric acid being taken, containing 44 grms., this is diluted to 400 c. c. and allowed to cool thoroughly. In taking the amount of cyanide, care must be taken to allow for the amount of impurity in the salt.

The solutions of potassium bromide, bromate, cyanide and of sulphuric acid are now mixed best by pouring the bromine salts and cyanide solutions simultaneously in a thin stream into a large funnel discharging into a bottle containing the sulphuric acid. It is essential to mix in this way otherwise reactions occur which occasion loss. The best way is to have the two solutions discharging simultaneously in a thin stream from large stop cock funnels.

After the mixing of the solutions the resultant solution should be agitated for about six hours to complete the reactions. The solution is then ready for use, and will contain about 4½ per cent of bromo cyanogen (theoretically in this example 4.75 per cent.)

Pure potassium bromate is a rather expensive salt, \$2.00 per pound if bought in small quantities (although in large quantities it can be had at about 50c per lb.), and in practice as mentioned further on commercial mixed bromine salts are used. For experimental purposes the mixed bromide and bromate salts can readily be made in the laboratory by the addition of liquid bromine to a saturated solution of potassium hydrate. This solution after the addition of bromine is heated until the bromine disappears, when more bromine is added, this being repeated until the red color stays permanently. The solution now contains a mixture of potassium bromide and bromate from which the bromate containing some bromide will separate out first when the solution cools, owing to its inferior solubility. This can readily be removed by filtration and the solution evaporated to the dry salts.

The reaction for the preparation of bromo cyanogen calls for a mixture of bromide and bromate, of which 41.2 per cent should be potassium bromate. If the solution containing the mixed salts made in the way described be evaporated down to dry salt, the mixture will contain considerably less than 4.12 per cent of bromate. For this reason it is desirable to remove successively the salt which settles out from the solution on evaporation and which is at first practically all bromate but gradually decreases in this salt until what precipitates finally is practically only bromide. In this way by having lots of salts, some high in bromate and others in bromide, the proper mixture of the two salts can readily be made especially if some extra potassium bromide, a common salt in the laboratory, be at hand.

The resultant solution should be neutral, as bromo cyanogen shows neutral to methyl orange and phenolphthalein, but may be slightly acid owing to the reactions being incomplete. If it should be acid, this acidity must be carefully neutralized by an N-10 potassium or sodium hydrate solution, care being taken not to add any excess, as this alkali decomposes bromo cyanogen, as already mentioned.

The bromo cyanogen solution prepared in this way is quite stable and keeps a considerable length of time, several months, if kept in a tightly stoppered bottle. It may turn a brown color upon standing, which color is *not* due to free bromine. This color does not interfere with the reactions.

Bromo cyanogen is made on a commercial scale, in Western Australia at Kalgoorlie as follows:*

* A. James, E. & M. Jour. Vol. 77. p. 31.

It is made from imported salts (Germany) which contain approximately 40 per cent of potassium bromate and the balance bromide. At Kalgoorlie to generate 100 lbs. of bromo cyanogen the following charge is used:

Mixed bromo salts,	125 lbs.
Cyanide, 100 per cent	65 "
Sulphuric acid, 70 per cent	147 "

The bromo cyanogen is made in a wooden, plain or lead lined vat of about 200 gals. capacity, securely covered with a lid through which a revolving paddle or stirrer works. Above this vat is a smaller vat in which is stored the necessary charge of potassium cyanide dissolved in 40 gals. of water. The mixing vat is first three-quarters filled with water, the agitator is started and the sulphuric acid added slowly and carefully. The whole is now left to stand and cool for one or two hours, as the heat generated by the addition of sulphuric acid would vaporize the bromo cyanogen were the rest of the ingredients added at once. When the contents of the vat are cool the mixed bromo salts are added gradually and, simultaneously the solution of cyanide is run in with constant stirring. The reaction commences immediately, but is not thoroughly completed until six hours of continuous agitation.

The resultant bromo cyanide solution is then added in the quantities desired to the agitation vats.

METHODS OF ESTIMATING THE AMOUNT OF BROMO CYANOGEN AND POTASSIUM BROMATE.

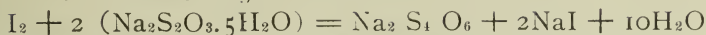
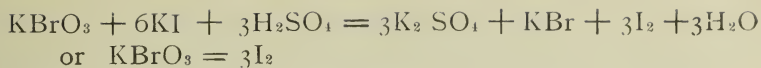
For the quantitative estimation of both potassium bromate and of bromo cyanogen an N-10 solution of sodium-thio-sulphate will answer. This solution will contain 12.4 grams of the salt. ($\text{Na}_2\text{S}_2\text{O}_3 \cdot 5\text{H}_2\text{O}$) and 1 c. c. will be equivalent to

0.00265 grms. of bromo cyanogen,
0.00142 " " potassium bromate.

For the Estimation of Potassium Bromate in the mixed salts of bromide and bromate proceed as follows:

Weigh out 200 milligrams of the dry salt, dissolve in 100 c. c. of distilled water, then add about 15 to 20 c. c. of dilute hydrochloric acid, and 3 to 5 grms. of potassium iodide. Iodine will be liberated and the solution is then titrated to colorlessness by N-10 sodium-thio-sulphate solution. If greater accuracy is required starch solution can be used as an indicator.

The reaction taking place is as follows:



For the Estimation of Bromo Cyanogen take 5 to 10 c. c. of the solution to be titrated, dilute with 25 to 50 c. c. of water, add 5 c. c. of dilute hydrochloric acid and 4 to 5 grams, of potassium iodide. This liberates iodine and the solution is titrated to colorlessness by N-10 sodium-thio-sulphate solution. Or as above, starch solution can be used as an indicator.

The reaction taking place is as follows:



In estimating mill solutions, or solutions that have been used on ores experimentally, and which contain potassium cyanide, the presence of this salt does not interfere with the titration for bromo cyanogen and in the estimation of potassium cyanide in these same solutions with silver nitrate in the usual way, bromo cyanogen does not interfere with the test except in so far as cyanogen, which may be present, due to the reactions between bromo cyanogen and potassium cyanide, interferes.

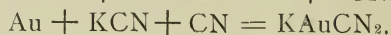
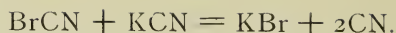
Usually the presence of cyanogen gives somewhat low results.†

* *METHOD OF MAKING BROMO CYANOGEN TESTS ON ORES.*—In experimental work the general practice is to add bromo cyanogen in solution to the test at intervals of 2 to 4 hours, the total amount of bromo cyanogen not to exceed one-fourth the amount of the potassium cyanide present in the experiment. As an example, suppose 16 assay tons of ore are treated with 700 c. c. of a 0.25 per cent, or 5 pounds per ton potassium cyanide solution, containing, therefore, 1.75 grams of cyanide. The total amount of bromo cyanogen to be added to this test is therefore 0.44 grams, and having a 3 per cent bromo cyanogen, solution each cc. containing .03 grms. the total number of c.c. to be added of this solution will be 15. If we make three additions at intervals of 4 hours, each we will add 5 c. c. each time.

The theory of the reactions with bromo cyanogen is as follows:

† James, Cyanide Practice, 1902, p. 140.

* James, " " 1902, chapter on Bromo Cyanogen.



The bromo cyanogen acts as a liberator of cyanogen thus practically taking the place of oxygen in the ordinary process. The nascent cyanogen acts as a hastener of the reaction and it is also probable that bromo cyanogen plus potassium cyanide may have a more powerful solvent action on certain gold minerals, such as calaverite, than the simple cyanide solution.

The idea of adding bromo cyanogen in small amounts from time to time is to have the liberation of cyanogen going on during the entire treatment. The greater the amount of potassium cyanide present in proportion to the bromo cyanogen the more rapid is the decomposition of the latter. With additions of bromo cyanogen as described in the test above, there will probably be no bromo cyanogen left in the solution after the expiration of 3 or 4 hours. Any bromo cyanogen left in solution after the expiration of the time of agitation is of course a waste and in tests following the amount may be cut down somewhat if this should be the case.

The consumption of potassium cyanide is greater with bromo cyanogen than without, and this must be taken into account in experimenting, not using too dilute solutions, especially in agitation tests in which consumption is abnormally high.

The solutions from the tests are very often a light brown or golden color, due probably to iron bromide, for more iron salts go into solution with bromo cyanogen than with plain cyanide. The earlier writers† dwelt at length on difficulties of precipitating the gold from the solutions when bromo cyanogen is employed, stating that bromo cyanogen has a decided action on zinc forming insoluble zinc bromo and zinc cyanide, which interferes with the precipitation. In the present state of the process there should be no bromide cyanide in the solution when it goes through the zinc, as it should be all decomposed. This difficulty in precipitation seems not to be encountered in West Australia, where bromo cyanogen is used extensively.‡

COST OF BROMO CYANOGEN.—The mixed salts of potassium bromide and bromate, containing approximately 40

†Gaze—Practical Cyanide Operations, 1898, p. 95.

‡For certain minor troubles of precipitation, see "Treatment of Telluride ores by Dry Crushing and Roasting at Kalgoorlie, W. A., W. E. Sampson., T. I. M. & M. Oct. 1903.

per cent of the latter salt can be readily obtained in this country* and can be delivered in 500 pound lots packed in barrels f.o.b. Deadwood at 25c per pound. Sulphuric acid costs at Deadwood from 2.6c to 3c per pound and potassium cyanide 23c per pound.

The cost per pound for chemicals, in making bromo cyanogen will therefore be, taking the Western Australia charge as a basis:

125 pounds mixed bromo salts at 24c per lb. =	\$30.00
65 pounds of potassium cyanide at 23c per lb. =	14 .95
147 pounds of 70 per cent sulphuric acid at 3c =	4.41
<hr/>	
Total cost 100 lbs. cyanogen	49.36
Cost per pound =	.50

The cost of labor, power, etc., of making may be estimated at 5c more, making a total cost of 55c per pound.

The following tables show concisely the results obtained and the details of the experiments on the blue siliceous ores of the Black Hills. The ores are taken from the various districts, Bald Mountain, Ruby Basin, Maitland and Yellow Creek, so as to give a representation of most of the blue ores.

*Can be bought from the Dow Chemical Company, at Midland, Michigan.

TABLE IV.

ORE "D"—A very dense blue ore, containing .22 oz. per ton, Tests made on 5 to 8 oz. lots, amount of solution 300 to 500 cc.

NUMBR.	Ore passed a Screen of the following mesh:	Strength of Cy- anide Solution. Pounds per ton.	Method of Treatment.	Amount of BrCN Added.	Time of Treatment.	Cyanide Con- sumption, lbs. per ton.	Assay of Tailings oz. per ton.	Per Cent of Extraction.
1	150 mesh	10	Roast at low heat plus plain cyanide.		24 hours.		0.11	50
2	150 mesh	10	Roast at high heat plus plain cyanide.		44 hrs. agitation; 48 hrs. standing.	2.2	0.08	63.6
3	150 mesh	10	Roasting at high heat plus cyanide plus bromo- cyanogen, treated for 12 hours before cyaniding with K ₂ H solution.	6 adds. of 4 cc. each of a 5 per cent sol.	Same.		0.08	63.6
4	150 mesh	10	Same.	6 adds. of 4 cc. each of a 5 per ct. sol.	Same.		0.06	72.8
5	150 mesh	10	Roasting at high heat, cyanide plus bromo- cyanogen.	Same.	Same.		0.07	68.9
6	150 mesh	10	Roasting at low heat, cyanide and bromo-cyanogen	4 adds. of 4 cc. each of a 5.5 per cent sol.	24 hrs. agitation.		0.09	59.0

TABLE I

ORE "A"—A dense blue silicious ore, containing 0.78 oz. gold per ton. For analysis, see previous pages. Tests made on 5 to 8 oz. lots, amount of solution 300 to 500cc.

Number.	Ore passed a Screen of the following mesh:	Strength of Cyanide Solution. Pounds per ton.	Method of Treatment.	Amount of BrCN Added.	Time of Treatment.	Cyanide Consumption in lbs. per ton.	Assay of Tailings, oz. per ton.	Percent of Extraction
1	20 mesh	20	Plain Cyanide, no alkali added.		36 hours agitation	4.9	0.52	83.3
2	40 mesh	20	Plain cyanide, no alkali added.		36 hours agitation	4.9	0.47	89.7
3	150 mesh	10	Plain cyanide, ore treated 12 hours with KOH. previous to cyaniding.		60 hours agitation	2.6	0.35	55.1
4	200 mesh	10	Same.		60 hours agitation	2.9	0.36	53.8
5	40 mesh	20	Plain cyanide, with addition of small amount of lead acetate.		60 hours agitation	12.7	0.41	47.4
6	80 mesh	20	Same.		41 hrs. agitation and 31 days standing	13.3	0.37	52.5
7	150 mesh	20	Same.		Same		0.36	53.8
8	40 mesh	15.1	Cyanide and bromo-cyanogen.	7 adds. of $\frac{1}{2}$ c.c. Ea. of a 3.5 per cent. sol.	Same	4.2	0.41	47.4
9	80 mesh	15.1	Same.	Same	22 $\frac{1}{2}$ hours agitation	4.7	0.40	46.1
10	150 mesh	15.1	Same.	Same	22 $\frac{1}{2}$ hours agitation	4.7	0.39	44.8
11	40 mesh	10.6	Same.	6 adds. of 4 c.c. ea. of a 3.5 per cent. sol.	22 $\frac{1}{2}$ hours agitation	7.5	0.42	48.7
12	80 mesh	10.6	Same.	Same	40 hours agitation	7.5	0.41	47.4
13	150 mesh	10.6	Same.	Same	40 hours agitation	7.5	0.40	46.1
14	40 mesh	20.0	Same.	0.98 grms. in 3 additions.	40 hours agitation	*14.0	0.39	44.8
15	80 mesh	20.0	Same.	Same	41 hours agitation	*13.8	0.41	47.4
16	150 mesh	20.0	Same.	Same	41 hours agitation	*10.5	0.37	52.5
17	150 mesh	10.6	Cyanide and bromo-cyanogen; ore treated with KOH. 12 hours before cyaniding.	6 adds. of 4 c.c. ea. of a 3.5 per cent. sol.	60 hours agitation	3.1	0.28	64.1
18	200 mesh	10.6	Same.	Same	60 hours agitation	4.2	0.30	61.5
19	150 mesh	8.9	Roasting, cyanide and bromo-cyanogen.	Same	48 hours agitation	5.1	0.10	87.1
20	150 mesh	10.0	Same.	4 adds. of 4 c.c. ea. of a 3.5 per cent. sol.	24 hours agitation	2.8	0.19	75.6
21	150 mesh	10.0	Same.	6 adds. of 4 c.c. ea. of a 3.5 per cent. sol.	44 hours agitation 48 hours standing		0.14	82.0
22	150 mesh	10.0	Roasting, cyanide & bromo cyanogen; ore treated with KOH for 12 hours before cyaniding.	Same	Same		0.11	85.8
23	150 mesh	10.0	Roasting and plain cyanide; ore treated with KOH, for 12 hours before cyaniding		Same		0.15	80.7
24	150 mesh	10.0	Roasting and plain cyanide.		Same		0.14	82.0
25	150 mesh	10.0	Same.		24 hours agitation	1.5	0.19	75.6
26	150 mesh	8.9	Same.		48 hours agitation	1.4	0.19	75.6

*Excessive consumption due to impure bromo-cyanogen.

TABLE II.

ORE "B." A dense blue ore, containing 0.90 oz. gold per ton.
Tests made on 2 to 5 oz. lots, amount of solution 200 to 400 cc.

For analysis see previous pages.

NUMBER.	Ore passed a Screen of the following mesh:	Strength of Cyanide Solution. Pounds per ton.	Method of Treatment	Amount of BrCN. Added.	Time of Treatment	Cyanide Consumption, lbs. per ton.	Assay of Tailings oz. per ton.	Per Cent of Extraction.
1	60 mesh	5	Plain cyanide, no alkali added.		24 hrs. agitation		0.47	47.7
2	150 mesh	5	Same.		Same		0.54	40.0
3	60 mesh	5	Plain cyanide.		13 days standing in bottle.		0.42	53.3
4	40 mesh	20	Plain cyanide, with addition of small amount of lead acetate.		41 hours agitation and days standing.	13.5	0.29	67.7
5	80 mesh	20	Same.		Same	11.0	0.27	70.0
6	150 mesh	20	Same.		Same	11.1	0.25	72.2
7	40 mesh	20	Cyanide and bromo-cyanogen.	0.78 grms. in 3 additions.	41 hours agitation.	8.2	0.34	62.2
8	80 mesh	20	Same.	Same.	Same	11.6	0.32	64.4
9	150 mesh	20	Same.	Same.	Same.	12.9	0.32	64.4
10	40 mesh	15.1	Same.	7 adds. of 2.4 cc each of a 3.5 per cent sol.	22½ hours agitation.	3.8	0.39	56.6
11	150 mesh	15.1	Same.	Same.	Same.	1.9	0.36	60.0
12	150 mesh	10.6	Same.	6 adds. of 4 cc each of a 3.8 per cent sol.	40 hours agitation.	5.1	0.37	58.9
13	150 mesh	8.9	Roasting plus plain cyanide.		48 hours agitation.	0.2	0.24	73.3
14	150 mesh	8.9	Roasting plus cyanide plus bromo-cyanogen.	6 adds. of 4 cc each of a 3.8 per cent sol.	48 hours agitation.	3.5	0.17	81.0
15	150 mesh	15.1	Roasting plus cyanide plus bromo-cyanogen treated with KOH. sol. before cyaniding.	7 adds. of 2.4 cc each of a 3.5 per cent sol.	22½ hours agitation.		0.21	76.6

TABLE III.

ORE "C." A dense blue ore, containing more sulphur than A. or B., containing 0.66 oz. of gold per ton. Tests made on 3 to 5 oz. luts, amount of solution 300 to 500 cc.

Number	Ore passed through screen of the following mesh:	Strength of Cyanide Solution, Pounds per ton.	Method of Treatment	Amount of Br. N. Added.	Time of Treatment	Cyanide Consumption, lbs. per ton.	Assay of Tailings oz. per ton.	Per Cent Extraction.
1	150 mesh	10.6	Plain cyanide, no alkali added.		40 hrs. agitation	8.4	0.32	51.5
2	150 mesh	10	Plain cyanide, treated with KOH solution for 12 hrs before cyaniding.		60 hrs. agitation.	2.9	0.33	50.0
3	150 mesh	10	Cyanide and bromo cyanogen, treated with KOH solution for 12 hours before cyaniding.	6 adds. of 4 cc. ea. of a 3.8 per cent. sol.	60 hrs. agitation.	3.7	0.28	57.5
4	150 mesh	15	Roasting plus plain cyanide, low heat for 4 hrs.		24 hours agitation.	2.0	0.24	63.5
5	150 mesh	10	Roasting high heat plus plain cyanide, treated for 12 hrs. before cyaniding with KOH.		44 hours agitation and 48 hours standing.		0.33	50.0
6	150 mesh	10	Roasting high heat plus plain cyanide.		Same		0.35	47.0
7	150 mesh	10	Roasting high heat, cyanide and bromo cyanogen treated for 12 hrs before cyaniding with KOH.	6 adds. of 4 cc. each of a 3.8 per cent. sol.	Same		0.27	59.0
8	150 mesh	10	Roasting high heat plus cyanide and bromo cyanogen.	Same.	Same		0.29	56.0
9	150 mesh	15	Low heat, same.	4 adds. of 2 cc. each of a 3.8 per cent. sol.	24 hours agitation.	7.0	0.20	69.5
10	150 mesh	8.9	Low heat, same.	6 adds. of 4 cc. each of a 3.8 per cent. sol.	48 hours agitation.	8.3	0.21	68.0
11	150 mesh	8.9	Roasting at low heat plus plain cyanide		48 hours agitation.	1.8	0.24	63.5

TABLE V.

Miscellaneous tests. In the main to show efficiency of bromo- cyanogen on certain ores. Tests made on 5 to 8 oz. lots, amount of solution 300 to 500 cc.

NUMBER.	Description of Ore.	Passed through a screen of Mesh.	Method of Treatment	Strength of Cyanide Solution pounds per ton.	Amount of BrCN Added.	Time of Treatment.	Cyanide Consumption, lbs. per ton.	Assay of Heads oz. per ton.	Percentage of Extraction.
1	Telluride ore, Victor mine, Victor, Colo.	150	Plain cyanide.	8.9		48 hrs agitation.	1.2	1.09	41.2
2	Same.	150	Cyanide plus bromo-cyanogen.	8.9	6 adds. of 4 cc. each of a 3.8 per cent sol.	Same.	4.7	1.09	91.7
3	Keystone concentrates pyrite, containing some arsenic.	table concentrates.	Plain cyanide treated with KOH. before cyaniding.	8.9		Same.	3.6	0.54	83.3
4	Same.	150	Same.	8.9		Same.	3.1	0.54	90.7
5	Same.	150	Cyanide and bromo-cyanogen, treated with KOH. before cyaniding.	10.6	4 adds. of 5 cc. each of a 3.8 per cent sol.	40 hrs. agitation.	6.4	0.40	100
6	Same.	150	Same.	10.6	Same.	40 hrs. agitation.	4.8	0.40	100
	Same.	150	Same.	8.9	6 adds. of 4 cc. each of a 3.8 per cent sol.	48 hrs agitation.	5	0.54	100
	Same.	table concentrates.	Same.	8.9	Same.	48 hrs. agitation.	5.5	0.54	100

SUMMARY OF EXPERIMENTS.—Influence of the fineness of crushing. The size of the ore is stated by "mesh" in the tables which means that the ore passed an ordinary assayers screen of the mesh stated. For further information the actual size of openings or space and size of wire of these screens is given.

Screen.	Number of Meshes per inch.	Decimal size of wire in inches.	Actual Opening between wires or space in inches.
	20	0.0165	0.0335
	30	0.01375	0.0195
	40	0.01025	0.01475
	80	0.00575	0.00675
	150	0.003	0.0033
	200	0.002	0.003

Material called 30 mesh was sized and gave the following results:

Per cent left on a 40 mesh screen	= 10.4	} coarser than 100 mesh 66.1 per cent.
" " " " 60 " "	= 29.8	
" " " " 80 " "	= 16.7	
" " " " 100 " "	= 9.2	
" " " " 150 " "	= 11.7	} finer than 100 mesh 33.8 per cent.
" " passed through a 150 " "	= 22.1	
Total	100.0	

In order to show how this product compares to actual mill products made in the Black Hills the following figures are appended:

PRODUCT OF A SIX FOOT MONADNOCK ROLLER MILL.—Screen, wire cloth with space of 0.046 inches, equivalent to 18 mesh.

Per cent left on a 20 mesh screen	= 1 per cent	} coarser than 100 mesh 31 per cent.
" " " " 40 " "	= 10 " "	
" " " " 60 " "	= 10 " "	
" " " " 100 " "	= 10 " "	
" " " " 200 " "	= 19 " "	} finer than 100 mesh 69 per cent
" " passed a 200 " "	= 50 " "	
Total	100. per cent.	

PRODUCT OF STAMPS.—Weight, 950 lbs. Screen 10 by 4 mesh No. 18 wire, space 0.053 inches. Dakota mill, Deadwood.

Per cent left on a 20 mesh screen	= 12.7	} coarser than 100 mesh 55.5 per cent.
" " " " 40 " "	= 22.0	
" " " " 80 " "	= 14.8	
" " " " 100 " "	= 6.0	
" " " " 150 " "	= 14.8	} finer than 100 44.6 per cent.
" " "passed thro' 150 " "	= 29.8	
Total	100.0	

PRODUCT OF DRY CRUSHING ROLLS.—Imperial mill, passed through a 16 mesh wire screen, 21 wire space .0305 inches.

Per cent left on a 20 mesh screen	= 3	} coarser than 100 mesh 64 per cent.
" " " " 30 " "	= 18	
" " " " 40 " "	= 17	
" " " " 60 " "	= 16	
" " " " 80 " "	= 5	
" " " " 100 " "	= 5	} finer than 100 mesh 36 per cent.
" " passed a 100 " "	= 36	
Total	100	

The effect of the size of ore on the extraction is shown as follows, the results being averages of tests made under same conditions:

Mesh. Laboratory Screen.	Ore A.	Ore B.
40	46.9 per cent.	62.2 per cent.
80	46.9 " "	64.4 " "
150	47.8 " "	64.4 " "

In order to test the effect of still finer crushing Ore A was passed through a 200 mesh screen, but the results were practically the same as with 150 mesh, a trifle lower even.

If, however, the ore is crushed so as to pass but a 20 mesh screen the extraction on all of the ores tested immediately fell off in such a manner that no further tests were made on such coarser sizes. The results of these tests are in accordance with the practice of the plants of the Black Hills, as will be seen by referring to the analyses of the mill products of several plants on the preceding pages. The product of rolls and screens at the Imperial mill is very close to that of a 30 mesh laboratory screen except that it contains a small percentage of somewhat coarser sands. The product of stamps at the Dakota mill, Deadwood,

contains a somewhat greater percentage of fines, (above 100 mesh) and less sands, but a considerable percentage of the sands is coarser than 40 mesh.

In the product from the Monadnock Roller mill the sands are considerable less in amount than in the product of the 30 mesh laboratory screen and the fines considerably more. This product is probably better suited for extraction than the stamp battery product above mentioned. In general it is true that while individual blue ores will differ somewhat in the fineness of crushing required, material that is coarser than 30 to 40 mesh (0.0195 to .01475 inches) will show an appreciable decrease in extraction. Product finer than 40 mesh (.01475 inches) will in general show but little better extraction, perhaps 2 to 3 per cent, than 40 mesh material. It seems, therefore, that for the greater portion of the denser silicious ores very fine crushing presents no advantage whatever, but that efforts directed toward keeping practically all of the sands finer than 30 mesh and preventing excessive sliming as an unnecessary evil is what is called for.

Tests that have been made on sand tailings at the Dakota mill show practically that all the sizes of sands, except the ones coarser than 30 mesh, have the same value.

THE EFFECT OF BROMO CYANOGEN.—The effect of bromo cyanogen while not what was hoped for is still very marked on all of the blue ores treated raw as the following table will show:

Ore.	Percentage of Extraction. Plain Cyanide.	Percentage of Extraction. Cyanide plus Bromo Cyanogen.	Increase in Extraction.
A	$\begin{cases} 54 \text{ per cent} \\ 39 \text{ per cent} \end{cases}$	$\begin{cases} 63 \text{ per cent} \\ 46 \text{ per cent} \end{cases}$	$\begin{cases} 7.0 \text{ per ct} \\ 7.1 \text{ per ct.} \end{cases}$
B	43 per cent	64 per cent	21.0
C	$\begin{cases} 51 \text{ per cent} \\ 50 \text{ per cent} \end{cases}$	$\begin{cases} 57.5 \text{ per cent} \\ 57.5 \text{ per cent} \end{cases}$	$\begin{cases} 6.5 \text{ per ct.} \\ 7.0 \text{ per ct.} \end{cases}$

The effect of bromo cyanogen on roasted ore is not so marked, but still strong in evidence as the following table will show:

Ore.	Percentage of Extraction by Roasting plus Plain Cyanide.	Percentage of Extraction Cyanide plus Bromo Cyanogen.	Increase in Extraction.
A	$\left\{ \begin{array}{l} 75.5 \text{ per cent} \\ 80.7 \text{ per cent} \\ 75.6 \text{ per cent} \\ 82.0 \text{ per cent} \end{array} \right.$	$\left\{ \begin{array}{l} 75.5 \text{ per cent} \\ 85.8 \text{ per cent} \\ 87.1 \text{ per cent} \\ 82.0 \text{ per cent} \end{array} \right.$	$\left\{ \begin{array}{l} \text{no} \\ 5.1 \text{ per ct.} \\ 11.5 \text{ per ct.} \\ \text{no} \end{array} \right.$
B	73.3 per cent	81.0 per cent	8 per cent
C	$\left\{ \begin{array}{l} 63.5 \text{ per cent} \\ 63.5 \text{ per cent} \\ 50.0 \text{ per cent} \end{array} \right.$	$\left\{ \begin{array}{l} 69.5 \text{ per cent} \\ 68.0 \text{ per cent} \\ 56.0 \text{ per cent} \end{array} \right.$	$\left\{ \begin{array}{l} 6 \text{ per ct.} \\ 4.5 \text{ per ct.} \\ 9.0 \text{ per ct.} \end{array} \right.$

The effect of bromo cyanogen on the siliceous ores of the Black Hills is to act merely as an accelerator in the solution of the gold bearing mineral as the following table shows:

Ore	Percentage of Extraction by Plain Cyanide, 3 days contact and 4 ¹ hours agitation.	Percentage of Extraction Cyanide and Bromo Cyanogen, 36 to 48 hours Agitation.	Per cent Ex. by Plain Cy. 24 to 36 hrs. agi.
A	53 per cent	48.7 per cent	39 per cent
B	71 per cent	61.4 per cent	43 per cent

The effect of treating the raw ores for a considerable period with a dilute caustic alkali solution before cyaniding has a marked effect on the extraction in some cases.

	Extraction by Plain Cyanide Without Alkali Treatment.	Extraction by Plain Cyanide and Alkali Treatment.	Ex. by Cyanide plus bromo cyanide plus alkali treat.
A	39 per cent	54 per cent	64 per cent

THE EFFECT OF ROASTING.—The effect of roasting on the ores is of course pronounced, but in some instances does not accomplish by any means what might be expected of it. Some of the ores in order to get extraction must be roasted with great care as regards temperature. The temperature must be kept low, not above a very dull red heat for 3 to 4 hours, when it may be raised to decompose any sulphates formed. If the temperature be raised to a bright red heat, early in the roasting, *with some ores*, but very little better extraction than on raw ore can be obtained. This is perhaps due to the presence of small amounts of arsenic and antimony that form stable insoluble compounds, locking up the gold and silver val-

ues. The effect of roasting can be seen from the following table:

Ore	Extraction by Plain Cyanide, Raw Ore.	Extraction by Plain Cyanide, roasted Ore.	Increase in Extraction.
A	$\left. \begin{array}{l} 39 \text{ per cent} \\ 33 \text{ per cent} \end{array} \right\}$	$\left. \begin{array}{l} 82 \text{ per cent} \\ 75 \text{ per cent} \end{array} \right\}$	$\left. \begin{array}{l} 43 \text{ per ct.} \\ 42 \text{ per ct.} \end{array} \right\}$
B	$\left. \begin{array}{l} 47 \text{ per cent} \\ 40 \text{ per cent} \end{array} \right\}$	$\left. \begin{array}{l} 76.6 \text{ per cent} \\ 73.3 \text{ per cent} \end{array} \right\}$	$\left. \begin{array}{l} 29.6 \text{ per ct} \\ 33.3 \text{ per ct} \end{array} \right\}$
C	$\left. \begin{array}{l} 50 \text{ per cent} \\ 51 \text{ per cent} \end{array} \right\}$	$\left. \begin{array}{l} 63.5 \text{ per ct.}^* 50 \text{ per ct.}^\dagger \\ 69.5 \text{ per ct.}^* 56 \text{ per ct.}^\dagger \end{array} \right\}$	$\left. \begin{array}{l} 13.5 \text{ per ct} \\ 18.5 \text{ per ct} \end{array} \right\}$

With ore B, roasting at a low heat for several hours, and then mixing with charcoal and re-roasting was tried, but with no better effect than before.

TIME REQUIRED FOR THE EXTRACTION OF THE VALUES.—Twenty-four hours continual agitation is about the minimum limit of time in order to get the extraction the ore will yield. Below that extraction will suffer materially. Increased time of agitation 36 to 48 hours, gives a somewhat increased extraction, but after that the increase of time gives but little increase of extraction, although of course a constant increment of increase is noted, * if the formation of soluble sulphides be avoided by the addition of a soluble lead salt or zinc sulphate in small quantities.

GENERAL.—A chlorination test on the ore B after roasting was made and gave an extraction of 75.5 per cent as compared to 81 per cent by roasting, cyanide and bromo cyanogen. Ore B, 150 mesh material, was also treated with a solution of aqua regia with boiling to see the effect and an extraction of 84.4 per cent was made, showing the exceeding refractoriness of a portion of the values in this class of ores.

From the experience gained in the tests made it seems that a portion of the values of the blue ores are practically insoluble in a cyanide solution, and that this insoluble compound is in many instances only partially altered by roasting.

If a portion of the values were locked up in the gangue of the ore so that it could not be extracted for this reason, it would seem

* Roasted at low heat.

† " at once at a high heat.

* See also Bulletin No. 3 South Dakota School of Mines. Cyanide experiments, by G. H. Clevenger and A. Forsyth.

that such fine crushing as 200 mesh would certainly liberate the greater part of it and thus permit a decided increase of extraction. But this is not so as can be plainly seen from the experiments. For these reasons the low extraction in most cases must be ascribed the non-solubility of a compound containing a portion of the values, and any effort toward finer crushing beyond the limits specified above, or a continued contact or leaching for a long time will be without results. However the time of treatment given the ores is a very important matter, as it must be continued long enough to be well above the minimum. There is also some evidence that when the time of treatment is continued for a considerable period of time that soluble sulphides tend to form which may lower the extraction unless some zinc be present in the solution to prevent their presence which, however fortunately, is usually the case in ordinary mill solutions.

METHOD OF APPLYING THE BROMO CYANOGEN TO THE MILL SCHEME OF THE WET CRUSHING PLANTS.—This is largely a matter of experiment, but the first addition might take place just before the solution enters the battery. A 3 to 5 per cent bromo cyanide solution could be introduced by a small pipe into the battery mains and regulated to furnish an amount of bromo cyanide solution so that the battery solution would contain, say, 0.01 per cent to 0.02 per cent of bromo cyanide. If added in this place the agitation given by the stamps could be taken advantage of. There is probably one very serious objection to the addition of bromo cyanide to the mill scheme as at present carried out and that is the alkalinity of solutions due to the lime added. As pointed out, alkalies destroy bromo cyanogen, probably without the formation of cyanogen, and it is very likely that the alkaline earth hydrates will do the same. It may be possible that if the battery solution entering the batteries carries but very little protective alkalinity and the amount of lime added with the ore be cut down as low as possible, something might still be accomplished with the bromo cyanogen.

This is a matter of experiment. In West Australia* at Kalgoorlie, lime is added after the agitation of the pulp with cyanide and bromo cyanogen solution, but as the solutions are re-

* The Diehl process, by H. Knutzen, Trans. Ins. M. and Met. June 19, 1902.

used must contain some lime when used with the bromo cyanogen.

The next place where an addition of bromo cyanide can be made is during the first addition of barren solution to the slimes. It could be added in amount so that the resultant solution in the slimes vat will contain from 0.01 to 0.02 per cent bromo cyanogen. If desirable another addition of bromo cyanogen solution may be made at a subsequent addition of barren solution.

The constant addition of bromo cyanogen to the battery solution would, however, require considerable bromo cyanogen per day and unless a rather marked increase of extraction, due to its addition, could be noticed, would hardly be profitable.

For example, in a plant crushing 125 tons of ore per day using 4 tons of a 2 pound battery solution, per ton of ore, the amount of bromo cyanogen required would be 120 pounds, in order to get 0.01 per cent of bromo cyanogen in the battery. This would cost \$66 exclusive of an increased consumption of potassium cyanide due to the addition of bromo cyanogen.

On \$8 ore, therefore, in order to make the addition of bromo cyanogen profitable, an increase of at least 7 per cent must be shown in the extraction, due to the addition of the bromocyanogen to the battery alone. The question would have to be worked out on a working scale in the mills themselves, and the cost of the experiment would not be very great, as experiments go.

The Crushing in Cyanide Solution Process as Carried on in the Black Hills of South Dakota.

BY CHARLES H. FULTON.

INTRODUCTORY.—The crushing in cyanide solution process was first introduced into the Black Hills at the old Dakota plant at Central City by Mr. John Hinton. The method originated in New Zealand, being first used by F. R. W. Daw in 1897, at the Crowns mine. In the Black Hills it has become practically the established method for the denser siliceous ores, there being at present five plants in operation using this method, with several more of the same kind projected. The dry crushing process still holds its own on the more porous and open siliceous ores and there are also plants in operation which do fine dry crushing, on dense siliceous ores. The mills employing the crushing in cyanide solution process are the Horse-shoe mill, 120 stamps, 60 in operation; the Dakota mill, 30 stamps; the Maitland mill, 40 stamps; the Hidden Fortune mill, 60 stamps, and the Lundborg, Dorr & Wilson mill, a six foot Monadnock Roller mill.

THE NATURE OF THE ORES TREATED.—For information on this subject reference is made to the foregoing paper.

GENERAL FEATURES OF THE PROCESS.

The process comprises the following operations:

1. The crushing of the ores, generally by stamps, in a cyanide solution ranging from 1.3 to 2.2 pounds of cyanide per ton, and carrying a protective alkalinity, equivalent to 1 to 1.5 pounds of sodium hydrate per ton.
2. The separation of the sands from the slimes by means of cone classifiers.
3. The treatment of the sands by percolation.
4. The treatment of the slimes by agitation and decantation.
5. The precipitation of the values by means of zinc thread.

The process is applicable to the dense siliceous ores that require a comparatively fine crushing and which contain but a small quantity of cyanide consuming compounds. For ores that

without previous alkaline treatment destroy much cyanide the process is not applicable. As a matter of fact it may be stated that the cyanide consumption in this method is higher than in dry crushing. The cyanide consumption in the wet crushing mills of the district varies from 0.75 to 1.50 pounds per ton of ore treated. At a typical dry crushing plant, the Imperial mill at Deadwood, milling the same class of ore, the consumption is 0.4 pound per ton. At the other dry crushing plants of the district it ranges from 0.4 to 0.75 pound per ton. The increased consumption of cyanide is a defect inherent in the process for several reasons. 1. Agitation of the ore with cyanide solution in the battery causes extra consumption. 2. Although the battery cyanide solution carries a protective alkalinity (alkalinity above that due to cyanide and cyanogen compounds) of from 1 to 1.5 pounds per ton, this does not by any means completely protect the cyanide from destruction by cyanicides. The reaction between cyanicides and cyanide and alkaline earth hydrates and caustic alkalis probably takes place in part at least simultaneously. It has been recognized by metallurgists that with many ores it is essential to apply a comparatively highly alkaline solution low in cyanide to the ores before the stronger cyanide solutions are employed, for the alkalinity carried in the strong cyanide solution would be ineffective in preventing a considerable consumption. 3. There is also an increase in the consumption due to the discharge of considerable cyanide in the moisture going out with the slimes tailings. This might be called a mechanical consumption. This consumption alone amounts to from 0.3 to 0.6 pounds per ton of ore treated. The mechanical consumption of dry crushing plants is but an insignificant factor.

At the present time it is difficult to make a comparison as regards the relative merits of the process under discussion and the dry crushing process. There is probably, on the whole, little difference between the two processes as regards cost although the wet crushing mills probably have a slight advantage in this respect, in spite of the slimes treatment and the higher consumption of chemicals. The wet crushing plants, of course, have an advantage in that they do not suffer from the dust nuisance. However, the hope that the wet crushing plants, on account of the great fineness of crushing that could be carried on, would be able to treat the blue ores in the raw state successfully, has not been verified and it is probable that for this class of ores roasting will finally have to be resorted to. In this case, of course,

dry crushing will have all the advantage. Some of the mines of the district furnish but little blue ore, while others have a great deal in their reserves. For the first type the crushing in cyanide solution method is, without doubt, a permanent institution.

THE CRUSHING OF THE ORES.—The ores are rough crushed generally by Gates crushers and in one instance, at the Maitland mill, by a Blake crusher. The crushed ore will pass a 1.5 to 2 inch ring and is fed to the stamps by Challenge feeds. Stamps with one exception are used for the fine crushing of the ore. The Lundberg, Dorr & Wilson mill employs a 6 foot Monadnock roller mill for the fine crushing, a set of rolls being placed between the Gates crusher and the roller mill, in order to get the proper sized feed of ore.

The following table gives the details of the stamps at four of the mills.

Details of the Stamp Mills.

Name of Mill	Weight of Stamp	No. of Drops per Min.	Height of Drop, Inches.	Depth of Discharge, In.	Width of Mortar Box at Discharge Level.	Screen Used	Amount of Solution per Ton of Ore.	Capacity per Stamp in tons per 24 hrs.	Type of Mortar.
Hidden Fortune*	1020	98	7	8	16	24 mesh, .6 wire	5	4	Double issue rear blocked up.
Maitland.....	910	97	7 to 8	6	13	26 by 13 mesh 26 wire.	4 to 5	3.5 to 4	Single issue.
Dakota, ..	950	88	8	9	15	10 by 4 mesh, 20 wire.	5	4	Single issue.
Dakota.....	950	88	8	7	22	Same.	5	4	Double issue rear blocked up.
Horseshoe.....	1000	90	8	5 to	18	14 by 7 mesh, 21 wire	6	4.5 to 5	Double issue rear blocked up.

*Crushing in very dilute cyanide solution and amalgamating with inside plate and over tables.

Some of the earlier mills installed double issue mortars with the idea of getting an increased stamp duty, but it was soon found that the amount of solution required in crushing was so great that the mills were unable to handle it economically and the rear discharges were closed by wooden frames. It will be seen from the table that the depth of issue and the width of the mortar at the discharge level vary considerably at the different mills. The weight of stamp has not so great a variation.

As the stamps used are for crushing purposes only at all of

the mills with the exception of the Hidden Fortune, it would seem that in general a narrow box, a shallow discharge and a heavy stamp, up to 1100 to 1200 pounds would give the greatest capacity and most economical results. However, the retaining of the ore in the mortar for a certain length of time in order to agitate thoroughly with cyanide solution is desirable. At most of the mills from 50 to 53 per cent of the values of the ore are extracted in the batteries and the classifying cones. Stamps for this type of crushing should attain a capacity of 5 tons and over per stamp, and in the later designs of mills to be built this has been provided for. At some of the mills the ore is very hard which in part accounts for the rather low capacities.

The screens used also vary considerably at the different mills, ranging from 26 mesh, 26 wire, to 10 mesh, 20 wire. This is necessitated by the requirements of the individual ores which, while having the same general characteristics, differ somewhat in the fineness of crushing required.

Recently several of the mills have installed wire cloth screens in which the opening is rectangular, instead of square, the long dimension in these screens being from 2 to 2.5 times that of the short dimension. Screens of this type give somewhat greater capacity and do not choke so readily as screens with a square mesh.

The denser siliceous ores require a comparatively fine crushing, but if the crushing is carried beyond a certain fineness nothing is gained in extraction and trouble is encountered in the production of an excessive amount of slimes which are difficult to handle in the mill. It has been demonstrated that if the ore is crushed so that the great bulk of it is not coarser than 30 mesh (0.0195 inches) and not finer than 60 mesh, (0.0075 inches,) the economic extraction is obtained. Material finer than 60 mesh yields but very little higher extraction than that between 30 and 60 mesh.

The 5 per cent greater extraction obtained in the mills on the slimes, although the recovery is the same as on the sands, is for the greater part due to the agitation obtained in the treatment. If the size of the ore particles however is coarser than 30 mesh the extraction on most of the ores is materially decreased.

The following table shows the nature of the mill product made at some of the mills:

Mechanical Analyses of Mill Products.

Percentage Remaining on Screen.	Name of Mill.			
	Mesh of Screen	Dakota Samps. Screen 10 by 4 mesh 20 wire.	Lundborg, Dorr & Wilson Monadnock Roller Mill, Screen 18 Mesh, 0.046 i space	Imperial, R. IIs. Screen 16 mesh, 21 wire.
	plus 20	12.7 per cent.	1.0 per cent.	3.0 per cent.
	plus 30			18.0 per cent.
	plus 40	22.0 per cent.	10.0 per cent.	17.0 per cent.
	plus 60		10.0 per cent.	16.0 per cent.
	plus 80	14.8 per cent.		5.0 per cent.
	plus 100	6.0 per cent.	10.0 per cent.	5.0 per cent.
	minus 100			36.0 per cent.
	plus 150	14.8 per cent.		
	minus 150	29.8 per cent.		
	plus 200		19.0 per cent.	
	minus 200		50.0 per cent.	

The Dakota mill uses the coarsest screens of any of the mills and gets a product about 20 per cent of which is coarser than 30 mesh, a rather high percentage, but in view of the very low tenor of the ores treated, and their shaly nature, this crushing is the most economic that could be practiced. The other mills use finer screens and their mill product approaches closely to that of the Monadnock mill quoted in the above table. One mill, the Hidden Fortune, crushes some cement ore which contains a considerable coarse free gold. This mill has adapted the crushing in cyanide solution process in conjunction with amalgamation inside and outside of the mortar. A very weak cyanide solution, 1.5 pounds per ton, is used with success as a battery solution, no difficulty being experienced to get good amalgamation. The plates need somewhat more frequent dressing owing to the hardening action of the cyanide on the

amalgam.* It is also very probable that the plates will have to be more frequently renewed owing to the solvent action of the cyanide.

WEARING PARTS OF THE STAMP MILLS.—Most of the mills have adopted chrome steel for shoes and in part for dies as giving the most satisfaction and being the most economical. Table III, compares the different materials that have been used.

The Cost of Shoes and Dies of Different Material.

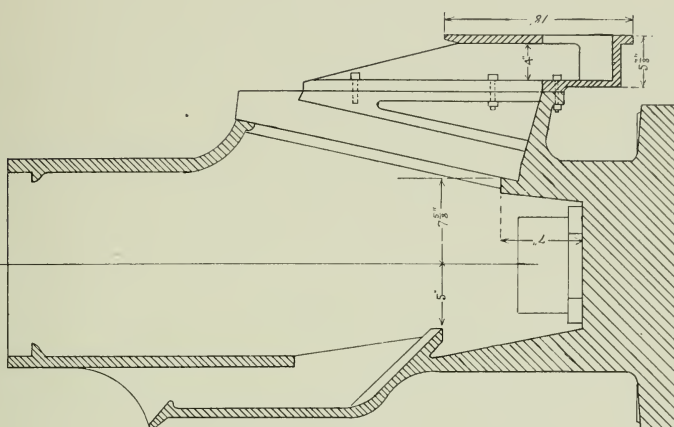
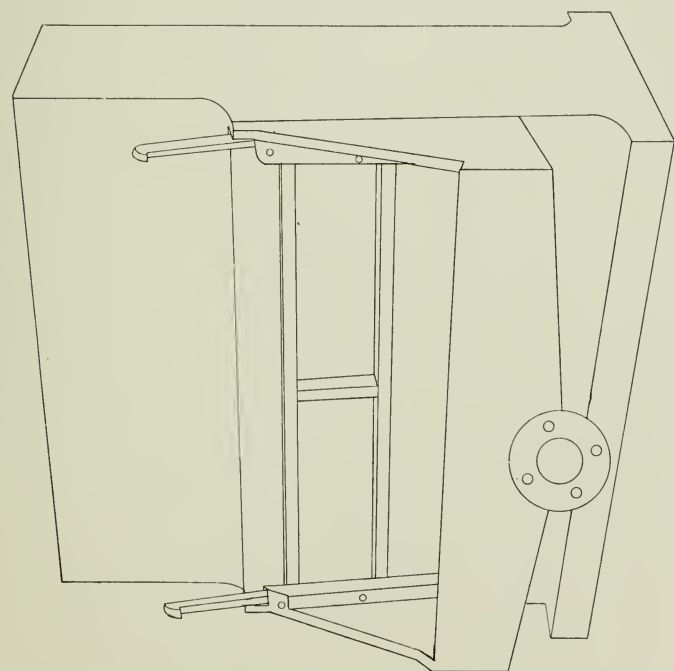
Note. Shoes weigh 180 pounds and dies 120 to 140 pounds. Laid down at Terry, S. D., chrome steel costs 5.83c per pound, Wilson forged steel, 5.72c per pound, and cast iron, 3.5c per pound.

Name of Mill.	Name of Part.	Material	Tons of Ore Crushed.	No. of Days Used.	Cost per Ton of Ore Crushed.
Maitland,	Shoe	Chrome steel	250	90-95	4.9c
"	"	Cast iron	105	35-4	4.45c
"	Die	Cast iron	105	40	3.28c
"	"	Wilson forged steel	280	105	3.06c
Horseshoe	Shoe	Chrome Steel	336	84	3.12c
"	"	Cast iron	104	26	6.15c
"	"	Wilson forged steel	280	70	3.67c
"	Die	Chrome Steel	400	100	2.29
"	"	Cast iron	120	30	4.08
"	"	Wilson forged steel	240	85	2.35

The Dakota mill also uses chrome steel shoes and is experimenting with a cast iron die containing 20 per cent of chrome steel scrap, made at a local foundry, which costs 3.5c per pound laid down at the mill. Dies of this kind weigh 120 pounds and lasted 46 days, crushing 175 tons of ore and leave 14 pounds of scrap which is sold at 0.5c per pound,

At the Lundborg, Dorr and Wilson mill at Terry a 6 foot Monadnock roller mill is used to crush in cyanide solution in

* This same method is used, in part at one or two mills employing the Diehl Process at Kalgoorlie, Australia. See the Diehl Process, H. Knutzen, Trans. I. M. and M. June 1902.



Mortar Box for Crushing in Cyanide Solution.

place of stamps. This mill crushes about 70 to 90 tons of ore per day, from 0.75 inch size through an 18 mesh screen, 0.046 inch space. The mill makes 32 revolutions per minute and has 19.5 square feet of screen area. A peculiar feature in the wear of this mill is that both the die ring and the roller tire cup on wearing, instead of the die cupping and tire crowning. This however does not seem to affect the efficiency of the crushing. The mill is giving satisfaction but no figures are as yet available to afford a comparison between it and the stamps on siliceous ore. It might be stated that the ores crushed at this mill are in part comparatively soft although some hard blue quartzite ores are also being crushed.

The cyanide solution is introduced into the batteries at most of the mills, by two 1.5 inch pipes entering at the front of the battery between the first and second and the third and fourth stamps. Each pipe is controlled by an iron cock.

At two of the mills a special form of mortar is used having a cast iron collecting launder bolted on at the front, and having a central discharge into the main launder collecting the sludge from all the batteries. This mortar is shown in the accompanying plate which also gives the form and dimensions of the mortar used at the Maitland mill. Generally all the screens are overhung with heavy canvas to avoid splash.

THE SEPARATION OF THE SANDS FROM THE SLIMES BY MEANS OF CONE CLASSIFIERS.—

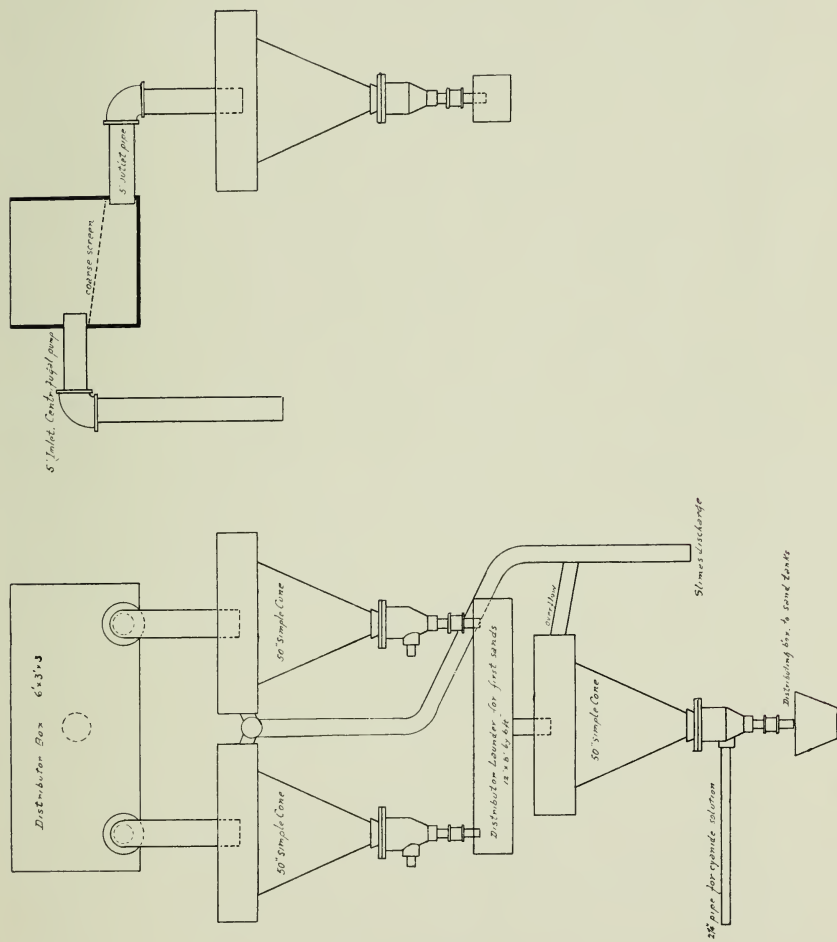
This is now done in the district almost entirely by means of simple sheet iron cones. These cones are the outer cones of ordinary hydraulic classifiers, the inner cones having been removed. It may be stated at the outset that the problem of removing the sands from the slimes when crushing in cyanide solution with considerable lime is a more difficult problem than when crushing in water with practically no lime. The lime causes much trouble, first, by its coagulating effect on the slimes, causing them to settle with the sands and coat sand particles with slimes, and second, by causing the formation of an excessive amount of froth or foam, which is certainly a great nuisance about the mill. The plate accompanying this part of the paper shows the general arrangement of the classifying cones.

The batteries discharge their sludge by launder into a central sump from which it is raised to the cones. The raising

pump at three of the mills is the Frenier Spiral sand pump, and at one of the mills a centrifugal pump. For the raising of the battery sludge consisting of sands and slimes the Frenier pump is preferred to a centrifugal on account of less wear. For the transference of slimes and for their agitation a centrifugal pump is generally used. The usual size of the spiral sand pump employed is the 54 by 10 inches. These pumps are run at 10 to 20 revolutions per minute, raising the pulp 15 to 20 feet. Twenty feet is about the practical maximum lift of these pumps and for greater lifts they are placed in tandem. A pump of the above size will readily handle from 350 to 450 tons of sludge per day.

The discharge of these sand pumps is intermittent so that at all of the mills a distributor box is used to steady the flow and give a uniform feed to the cones. These distributing boxes have different forms at the various mills. At the Horseshoe mill a pyramidal box is used 4 by 4 feet in cross section at the top, the sides sloping at 60 degrees to meet at a point. The inverted pyramid is topped by a box 12 inches high through which the two 4 inch pipes from the sand pumps enter. About 12 inches from the bottom of the pyramid four three inch pipes emerge one at each side, which feed into four 50 inch cones. The distributor is placed centrally over the four cones and as low as possible so that the head under which the discharge takes place will be small. A screen placed in the distributor box serves to keep out foreign matter from the cones. At the Maitland mill a plate steel box 6 by 3 feet in cross section and 3 feet deep is used as a distributor. On one side 21 inches from the top two 5 inch pipes enter from the sand pumps and discharge upon an inclined screen. The two discharge pipes 5 inches in diameter which feed the two 50 inch cones have their centers placed 4 inches above the bottom of the distributor box. As at the Horseshoe mill the distributor is set as closely as possible to the cones. At the Dakota mill a similar box made of wood is used.

The upper cones are simple cones of sheet iron from 40 to 50 inches in diameter having vertical sides at the top 12 inches high. The slope of the cones is 60 degrees ending in a six inch sorting column, which has a two inch discharge controlled by an iron cock. The charging pipe feeds at the center of the cone just below the pulp level. In most of the mills the top cones are covered closely by either a wood or an iron



Apparatus for the Separation of Slimes from Sands.

cover to confine the foam. This has the disadvantage, not very serious, of preventing ready inspection of the cones.

The upper cones are practically simple settling cones. The sludge going to the cones contains from 14 to 19 per cent of solids of which 30 to 50 per cent is slimes and the rest sands. Just what constitutes sands and slimes is somewhat difficult to define.* It is rather generally accepted by the men in charge of the plants, crushing siliceous ores, that material finer than 150 mesh is a slime and coarser than 150 mesh a sand, and classification is made largely on this basis. It has also been defined as that portion of the crushed ore that will make water muddy, sands, no matter how fine, settling practically at once and not remaining suspended.†

The overflow from the upper cones contains practically no sands, even very fine, and goes to the slimes tanks by the overflow launder. The sands discharged at the bottom of the cones contain from 20 to 35 per cent of slimes and are distributed by a short box to the lower cones. These lower cones are of the same construction as the upper ones, but have introduced into the sorting column an upward current of cyanide solution either battery solution or barren solution (solution that has been precipitated), but generally battery solution. This solution is introduced through a 2 inch pipe with a cock to regulate the flow. The amount of solution introduced in this way amounts to from 60 to 80 tons per 24 hours for a 42 to 50 inch cone. These figures vary somewhat, those given representing the limits. The number of lower cones is always one-half that of the upper cones. The solution pipes entering the sorting column of the lower cones do not come directly from the stock tanks, but from a special box provided with an overflow at a definite height, so that the head of the entering solution is always constant.

The final sand discharge containing 25 to 30 per cent solids, and containing from 1 to 5 per cent of slimes, goes to the Butters distributors over the sand vats, battery solution being added in the carrying launder so as to have 5 parts of solution to 1 part of sand. The mills endeavor to make a fairly close separation of sands from slimes in order to get a good leaching rate in the sand tanks, usually from 2.5 to 3.5 inches

*What constitutes a slime? W. J. Sharwood, E. & M. Jour. vol. 76, p. 539, 650.

†Thus defined by Mr. John Gross, in a paper read before the Black Hills Mining Men's Ass. "Cyanide Practice at the Maitland Properties."

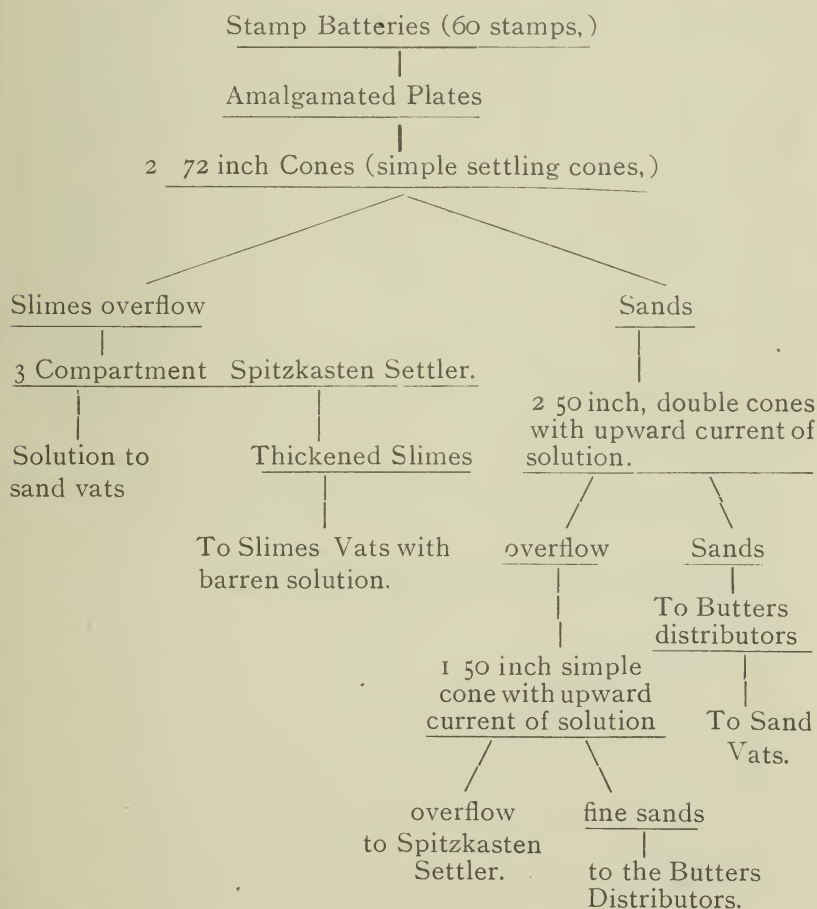
per hour, although at one plant it is but 1.5 to 1.75 inches per hour and also to prevent trouble which the sands give in the slimes tanks, that of settling to the bottom and remaining there during the greater part of the treatment practically unacted upon..

A very close and satisfactory separation however is not possible, first on account of the inherent defects of the cones used as classifiers and second because of the bad effect of the lime in sending slimes with the sands as already mentioned. For these reasons the classification adopted is that of making a clean sand rather than a clean slime, this being the lesser of two evils. For example at the Maitland mill the sands carry only one to two per cent of slimes, five per cent giving an unsatisfactory leaching rate. In making sands of this kind the slimes run from 15 to 20 per cent of fine sands, but a small portion of which remains on a 150 mesh screen. The proportion of the ore crushed treated as sands and slimes varies at the different mills. At the Maitland mill the average figures for 8 months show 48.2 per cent of the ore treated as sands and 51.8 per cent treated as slimes. At the Dakota mill the sands amount to 65 to 70 per cent and the slimes to 30 to 35 per cent. At the Lundborg, Dorr and Wilson mill the sands and slimes amount to approximately 50 per cent in each case. At the Horseshoe mill the slimes amount to 26 to 30 per cent and the sands to 70 to 74 per cent.

A number of different systems of classification by the cones were tried before the system described was adopted. It will be noticed that the system now used reclassifies the sands from the upper cones. Formerly the plan was to reclassify the slimes overflow from the upper cones in the lower cones, but this practice was soon discarded as unsatisfactory, giving in some instances unleachable sands. Double cones were also used, i. e. the regulation cone classifier, but most of the mills now classify with the inner cone removed. The only mill where a double cone with an upward current is used to reclassify the sands is at the Hidden Fortune mill.

Two of the mills, the Lundborg, Dorr & Wilson and the Hidden Fortune, unwater the slimes before they go to the slimes tanks. The first by means of a large sheet iron cone 22 feet in diameter, the top portion sloping 40 degrees and the lower portion near the discharge 60 degrees, and the second by means of a 3 compartment spitzkasten 40 feet long, 6 feet wide and 8 feet deep. The compartments are charged successively and

the thickened slimes drawn off and mixed with solution in the launder that transfers them to the slimes tanks. The object of unwatering the slimes in this way is to give them an additional treatment with barren solution, for when not unwatering the slimes they go to the slimes vat with battery solution and are settled there for the first time, while with the unwatering device the slimes go to the slimes tanks with barren solution having had one dilution by the time they reach the first slimes tank. The scheme of classification at the Hidden Fortune mill is given below as it is somewhat different from that of the other mills:



To show the nature of the classification at some of the mills the following mechanical analyses of sands and slimes is appended:

MECHANICAL ANALYSES OF SANDS AND SLIMES.

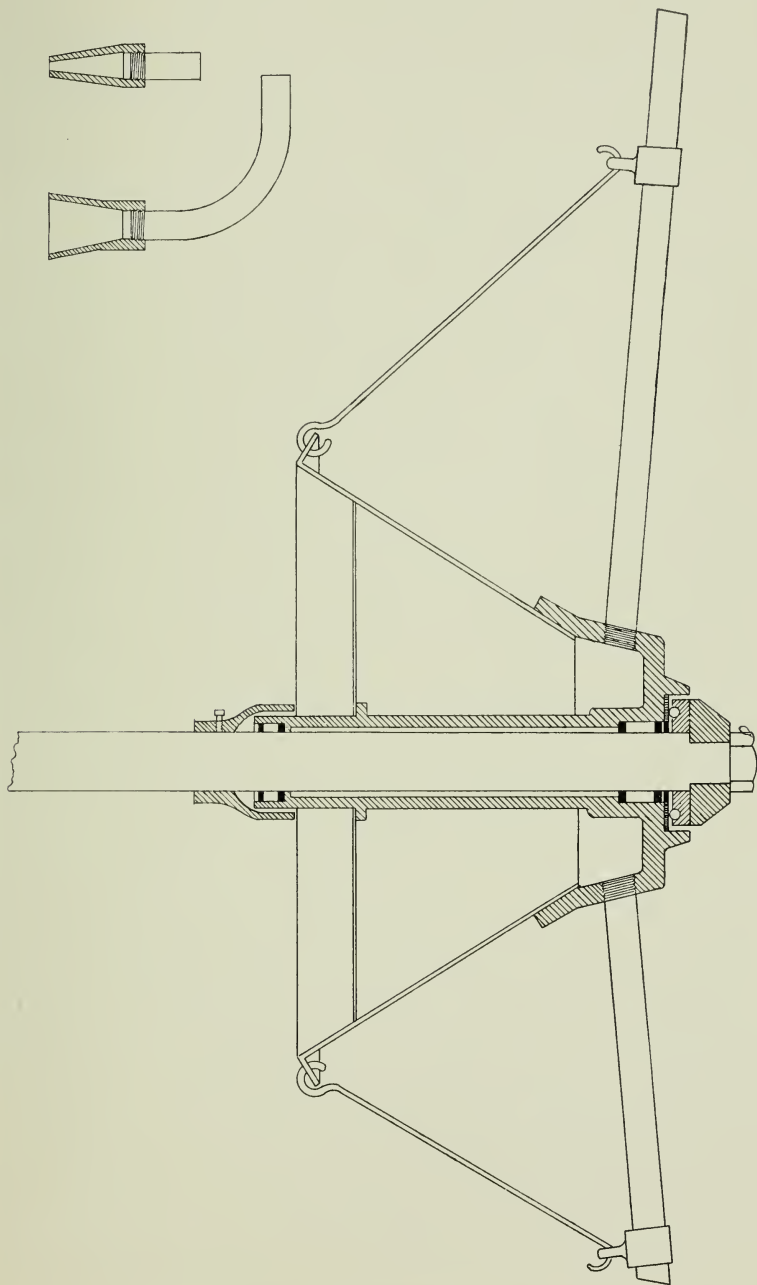
Sands at the Dakota mill constituting 70 per cent of the mill product	Slimes at the Dakota mill constituting 30 per cent of the mill product.
On a 20 mesh screen 13 to 20 per cent.	On a 100 mesh screen 0.3 to 0.4 per cent.
" " 40 " " 30 " "	On a 150 mesh screen 12 to 33 per cent.
" " 80 " " 26 " 54 " "	Passed a 150 mesh. 60 to 87 " "
" " 100 " " 7 " 8 " "	
" " 150 " " 13 " 18 " "	
Passed a 150 mesh 4 " 5 " "	
Sands at the Lundborg, Dorr* and Wilson mill, constituting 50 per cent of the mill product.	Slimes at the same mill, constituting 50 per cent of the mill product.
On a 40 mesh screen 30 per cent.	On a 60 mesh screen 0.5 per cent.
" " 100 " " 40 " "	" " 100 " " 1.5 " "
" " 200 " " 24 " "	" " 200 " " 18 " "
Passed a 200 mesh 6 " "	Passed a 200 mesh screen 80 " "

It may be noted that a comparison of these products at the different mills is not possible as the ores differ and what is a fine sand at one mill according to mesh size may be a slime at another.

The proper separation of the sands from the slimes is a vital question to be solved with the plants of the Black Hills and is one that has given the mill men much trouble. While the present system is a great improvement on the practice of the earlier mills there is still much room for further improvement.

THE TREATMENT OF THE SANDS.—The filling of the sand tanks is accomplished by distributors of the Butters and Mein type, the construction of which is shown in some detail in the accompanying plate. The distributor is suspended from a trolley running on tracks above the sand vats, so that the distributor can readily be transferred from one vat to the other. The sands are fed into the hopper of the distributor by a launder, which feeds as near the center of the hopper as possible, avoiding the throwing of the feed against the sides as this causes an irregular distribution of the sands in the vat. The dimensions of the distributors vary according to the capacity required. The slope of the pipe arms is 1 in 12, and the diameter of the pipes varies in the different distributors from 1.5 to 2.5 inches. Generally all the pipe arms in a distributor are of the same diameter, but in the one at the Horseshoe mill the long arms are 3.5 inches, the medium arms are 2.5 to 3 inches and the short arms 2 inches in diameter. The discharge nozzles are usually separate castings, the discharge being controlled by wooden

* This mill has replaced cone classifiers, by a mechanical classifier, the invention of Mr. J. V. N. Dorr.



SCALE 1:10

Distributor for Sands.

plugs. The number of arms is generally six, although the distributor at the Horseshoe mill has 8 arms. In the case of one of the six arm distributors the following figures give the length of arms, 13.25 feet, 11.5 feet, 9.5 feet, 8.0 feet, 5.5 feet and 2.5 feet. These arms are unsymmetrically hung in such a way that their weight balances the distributor. The discharge of the pipe arms must cover the surface of the vat. The hoppers of the distributors are provided with a horizontal screen to keep foreign matter out of the pipe arms. The function of the distributors in the mills crushing siliceous ore is not in part that of a classifier acting with a filled vat in removing slimes from sands, but it acts solely to evenly distribute sand in the vats. The sands are not laid down under water or solution, but the vat is what might be called dry filled, the solution which goes into the vats continually draining off through the filter until the vat is full of sands. The top layer of sands in the vat is always practically dry. This method of filling has the advantage, first, that the slimes in the pulp are uniformly distributed with the sands in the vat, which is not the case when direct filling is employed under water; second, that for this reason it gives a charge that is more percolable, and third, that during the filling a great amount of solution passes through the sands, in this way treatment going on all the time that the vat is filling. The charge laid down in this way is also more porous than when laid down under water. At the Maitland mill the amount of solution passing through a 150 ton charge of sands while filling is 700 tons, or 4.7 tons of solution per ton of sands.

The time of filling a 30 by 6 foot vat at the Maitland mill is 60 hours; at the Lundborg, Dorr & Wilson mill a vat, 18 by 10 feet is filled in 60 to 72 hours. At the Dakota mill a 115 ton vat is filled in 38 hours.

The method of filling formerly employed at the first wet crushing plants of the Hills, the Portland and the Dakota Mills, was the indirect method, settling boxes with two compartments being used, these compartments alternately discharging their contents into the sand vats below, where the charges were raked over and leveled off. At the Dakota mill double treatment of the sands was also resorted to, but discarded as unnecessary after a year's trial. The settling boxes were found to be such inefficient classifiers that the cone system described was evolved and the sands charged at some of the mills, by distributors into the sand vats filled with solution. All of the plants however soon adopted the method of "dry filling" described as

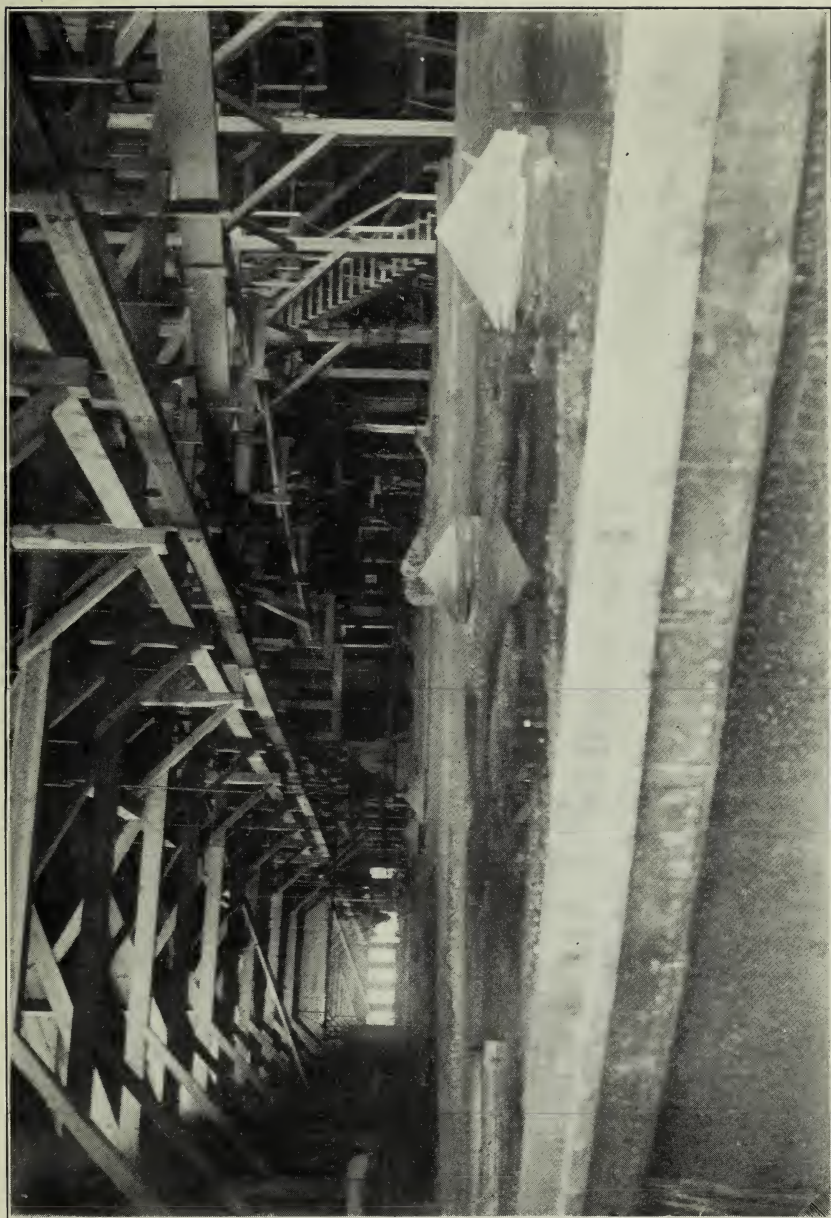
more satisfactory.

The general method of the treatment of the sands is the same at all of the mills although the amount of solutions and the time of treatment varies. The treatment of the sands is determined as far as extraction will permit by the problem of handling the mill solutions, which in a plant of the type under discussion is quite complex as might be expected.

The following table shows some of the details of sand treatment:

Name of Mill.	Capacity of Tank.	Amount of solution passing while filling.	Amount of Battery Solution.	Amount of Barren Solution.	Amount of Wash Water	Total Time, Days.
Maitland.....	140 tons	700 tons	900 tons	450 tons	15 tons	16
Dakota.....	115 tons	86 tons		63 tons	20 tons	5
Horseshoe.....	350 tons	400 tons				8

Solutions are leaching through the sands continually, there being no "contact" or solution standing on the ore as in dry crushing mills. There is also no strong solution properly so called, although the battery solution and the barren solution differ slightly in strength, in some mills the barren solution being the stronger while in others the battery solution is the stronger. At the Maitland mill the battery solution carries 1.20 to 1.30 pounds of cyanide per ton, and the barren solution 1.50 to 1.60 pounds per ton. At the Horseshoe mill the battery solution carries 1.4 pound of cyanide per ton and the barren solution is somewhat stronger, though of indefinite strength. At the Horseshoe mill, the overflow solution from the slimes vats while these are filling, is standardized in a sump tank up to three to four pounds of cyanide and then run through the sands. At the Dakota mill the battery solution contains 2.2 pounds of cyanide per ton, and the barren solution two pounds per ton. At the Lundborg, Dorr & Wilson mill the battery solution contains two pounds of cyanide per ton and at the Hidden Fortune mill it contains 1.3 pounds per ton. The amount of wash water varies but little at the different mills amounting to 0.1 to 0.2 tons per ton of sand. Little wash water is required as the cyanide solutions are all weak and such large amounts of solution are passed through the sands in most of the mills. The deficit of solutions in the



Sand Vats, Mogul Mill, Horseshoe Mining Company.

mills is made up mainly from wash water added in the slimes treatment.

The following figures show the result on sands obtained at the Dakota mill, over a period of 5.50 months: The average value of the ore was \$4.75 per ton. The sand tailings averaged \$1.22 per ton. This gives an extraction of 74.25 per cent on the sands. The moisture going out with the sand tailings had a value of 40 cents per ton. During May, 1904, the average value of the ore was \$4.55 per ton. The average value of the sand heads as charged into the vats was \$2.60 per ton, the average value of the sand tails unwashed was \$1.06 per ton giving an extraction of 76.7 per cent on the sands. Comparing the original value of the ore, the sand heads and the sand tails, it is evident that 42.8 per cent of the extraction takes place in the batteries and cones, and 33.9 per cent during the sand treatment proper.

At the Hidden Fortune mill the extraction on the sands averages 75 per cent. For the extraction on the slimes and the total extraction reference is made to the figures given under slimes treatment.

THE TREATMENT OF THE SLIMES BY AGITATION AND DECANTATION.—There are two systems of slimes treatment practiced in the Hills.

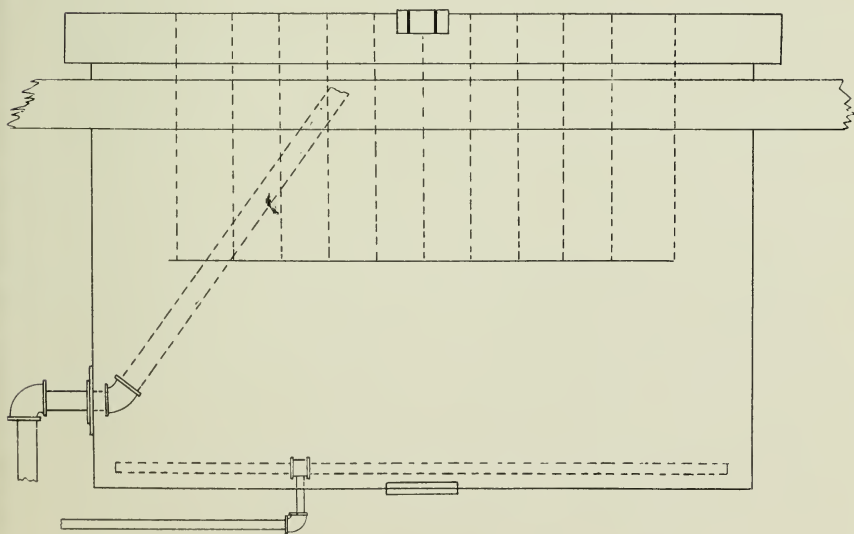
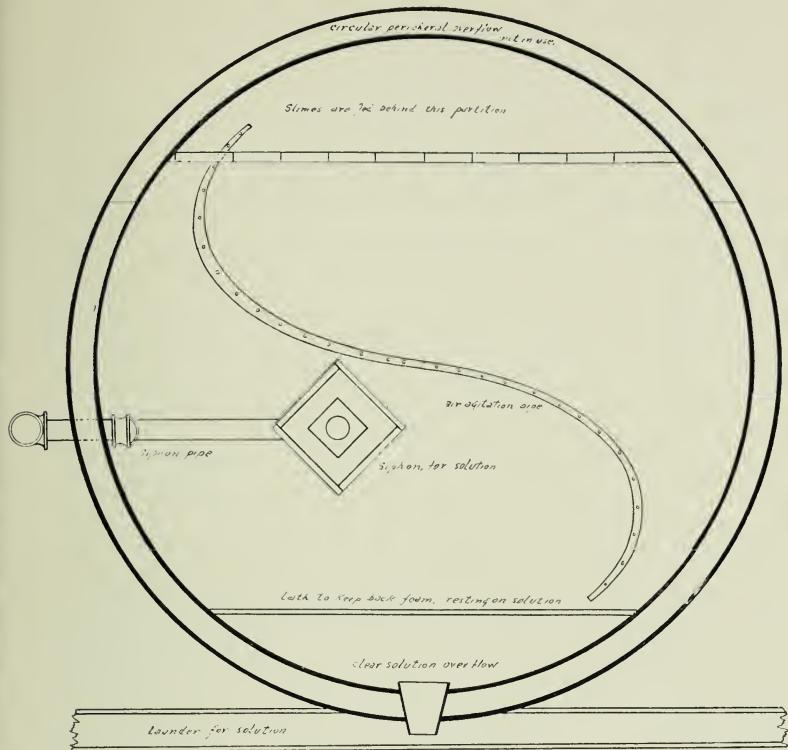
1. That in which the treatment of the slimes is completed in the vat into which they are originally charged, and in which most of the agitation is performed by compressed air.

2. That in which the slimes are successively transferred from one vat to another, there being generally three to four transfers before the slimes are discharged. The agitation in this case is done by means of centrifugal pumps.

The first method is practiced at the Horseshoe mill as follows: there are 16 slimes tanks, 14 feet in diameter and 10 feet deep and two 30 feet diameter and 16 feet deep. The arrangement of a slimes tank as shown in the accompanying plate. A partition curtain runs down to nearly the bottom at one side of the tank behind which the slimes are charged as they come from the cones. Before charging slimes the vat is filled with barren solution, then the slimes are run in, the surplus solution running off clear at the lip, any foam being held back by a strip of wood or lath resting on the surface of the solution. Four to six pounds of lime are added per ton of ore at the batteries for the coagulation of the slimes, this being the only addition of lime made in the mill. The addition of lime must be some-

what nicely adjusted as too little lime fails to coagulate the slimes readily and too much gives trouble in the precipitation of the values later on. The slimes settle rapidly and the solution usually runs off clear at the lip until the slimes have accumulated to the extent of about 50 inches, equivalent to about 25 to 30 tons of dry slimes. When the solution at the lip becomes cloudy the slimes charge is turned into the next tank and the slimes in the tank just filled are permitted to settle. This settling takes about ten hours. While the slimes are settling the supernatant solution is decanted off by means of a decanting device which is a simple wooden frame with a pipe at the center which is connected with a take-off pipe about 18 inches from the bottom of the vat. (See plate.) It is important to permit the slimes to settle as low as they possibly can and to decant as closely as possible without taking any of the muddy solution. The object of the slimes treatment in the main is to remove by successive dilutions the dissolved values, so that it is evident, unless the decantation is as close as possible each time, it partly fails in its object. The solution can usually be decanted within an inch of the settled slimes. When the decantation is complete a wash of barren solution is added, amounting generally to 40 tons and during the addition of this the charge is agitated by compressed air at 40 pounds pressure per square inch. The air is introduced on the bottom of the vat through two S shaped pipes crossing each other, and having 0.12 inch perforations. The only agitation the slimes receive is that obtained by the air. The agitation by air alone is a weak point in that it fails to move all of the material, especially the heavier portion of the slimes and the fine sands at the bottom of the vat. For that reason on discharging a vat, it is not sluiced out, but what slimes will run out by the bottom gates are let go, and the heavy thick slimes remaining, amounting to two to four tons per charge, form again a portion of the next charge thus getting two treatments. Each charge of slimes gets four to six washes with barren solution and one wash with water, each wash amounting to 40 tons. This gives 0.5 tons of barren solution and 1 to 1.6 tons of wash water for each ton of dry slimes treated.

The slimes as discharged contain very close to 50 per cent moisture. From ore averaging \$8 to \$9 per ton dried slimes tailings average \$1.75 per ton, 30 to 40 per cent of which still existed as gold in solution. This gives the washed slimes tail-



Slimes Vat and Accessories.

ings a value of \$1.24 per ton and the solution discharged as moisture, a value of 50 cents per ton.

In the treatment of slimes by successive dilutions for the extraction of the values where, in the total treatment a definite amount of solution is used per ton of dry slimes, it is theoretically required, in order to get the maxim extraction, to use the amount of solution in a comparatively large number of dilutions of small amount each time, rather than a few dilutions of large amount each time. Thus, in treating a ton of slimes with six tons of solution, it is theoretically better to give six dilutions of one ton each, rather than two dilutions of three tons each. If any one dilution is larger than the other it should, of course, be applied when the slimes are highest in value. It must be borne in mind however, that in the slimes treatment there is a solution of values constantly going on during the treatment so that while the solution is being reduced in value by dilution it is constantly being augmented by the solution of new values, so that the solution finally discharged as moisture with the slimes tails will never be as low in value as the dilution calls for. Since the cost of applying a dilution of definite quantity to the slimes is the same, no matter what the value extracted by the dilution is, it will be seen that the economic limit is soon reached where further dilution will not pay. Few of the mills can afford to apply more than four to five dilutions profitably. It must also be borne in mind that with an increased number of dilutions the amount of solutions to be handled in the mill will increase. As it is, at present the mill handles a great quantity of solution per day. The application of an extra water wash to the slimes would, however, not be so objectionable if the decantations from the last wash were run to waste through a large zinc box in which the poorer grade of shavings and the dust from the lathe could be utilized. The saving made in this way would probably be appreciable.

The treatment of the slimes at the Hidden Fortune mill is similar, but differs in details. The slimes vats are first filled with barren solution and the slimes pulp is charged at the center of the vat through a large pipe which leads to within a few feet of the bottom of the vat. The clear solution overflows continually around the whole of the periphery being collected by an annular launder and taken to the battery sumps. When the outgoing solution becomes cloudy the charging of the slimes is stopped and slimes are permitted to settle the clear solution at

the top, in the mean-time being decanted to the battery sump, in a similar way described for the Horseshoe mill. When the solution has been decanted to within an inch or two of the slimes the top layer of thin slimes, to a depth of three or four inches, is pumped to the vat that is filling with slimes, the object of this being to remove as much solution as possible in order to have the next dilution as efficient as possible. The first wash of barren solution is then added, this being added through the perforated air pipes at the bottom of the vat. This method of adding the barren solution is adopted, first, to keep the perforations of the air pipes clear, and second, to secure the agitation and moving of the heavier slimes at the bottom of the vat. After the addition of barren solution the charge of slimes is agitated with air at 40 pounds pressure per square inch, then permitted to settle and the clear solution decanted as described.

The slimes receive three washes with barren solution and one with water. Before the slimes are finally discharged the top layer, to a depth of three or four inches, is again pumped to the vat which is filling. The total time required for the treatment of the slimes is between three and four days. Three washes are found sufficient at the Hidden Fortune mill since the slimes receive an unwatering before going to the slimes vats, as has been described. At this mill the extraction made on slimes as determined by the assays on the washed tailings is 80 per cent. The actual recovery is but 75 per cent, as determined by the unwashed tailings, showing a loss of gold of five per cent which goes out with the slimes tailings in the dissolved form.

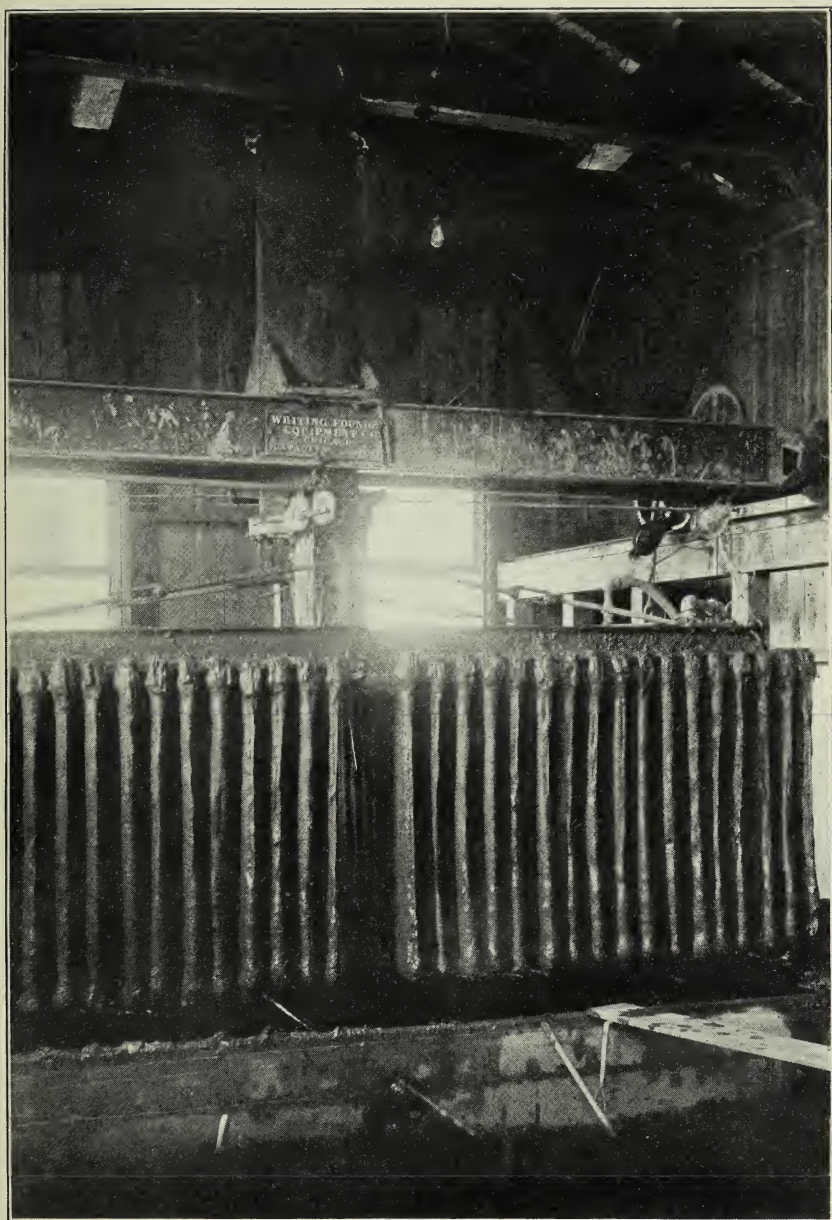
The second method of treatment in which the slimes are successively transferred from one vat to another is illustrated by the practice at the Dakota mill. The slimes from the cones are alternately charged into the loading vats, No's. 1 and 2, which are 20 feet in diameter and 137 inches deep. Each vat is filled for twelve hours then permitted to settle for ten hours, the clear solution at the top being continually decanted off, finally to within one inch of the settled slimes. The settled slimes are then pumped by a centrifugal pump having a four inch suction to vat No. 3, barren solution being continually added to the suction of the pump during the transference which takes from one to three hours. In vat No. 3 the slimes are permitted to settle again for ten hours, the clear supernatant solution being decanted off meanwhile. The slimes are then transferred to vat No. 4 by the pump, barren solution being added to the suction. In vat No. 4 the slimes receive an additional agitation by pump-

ing for about one hour, the slimes being drawn off at the bottom of the vat by the pump and returned over the top. The settling and decanting is then repeated and the slimes transferred to vat No. 5. From this vat after the settling and the decantation of solution the slimes are transferred to vat No. 6, but this time with wash water instead of barren solution. The amount of wash water added is equivalent to the amount of moisture in the slimes. In vat No. 6 the last settling takes place, the solution is decanted and the slimes are discharged. Lime for the coagulation of the slimes is added to the extent of six pounds per ton of ore at the batteries. The time required for the treatment of the slimes is five days.

The following figures on slimes treatment at the Dakota mill are of considerable interest: In 5.5 months 6,681.67 tons of slimes were treated, the slimes amounting to 33 per cent of the ores crushed. The average value of ore during this period was \$4 75 per ton. The slimes tailings dried and unwashed assayed \$1.31 per ton, giving a recovery on the slimes of 72.28 per cent. The washed slimes tailings assayed \$0.912, showing a solution of the values of 80.7 per cent and a loss of soluble gold of 40 cents per ton or 8.42 per cent of the value of the ore. The slimes are discharged with 50 per cent moisture, this moisture consisting of solution having a strength of 1.07 pounds of cyanide per ton. Since there is a ton of this solution going to waste for every ton of dry slimes discharged, in the 5.5 months which the above period covers, there were lost 7,147.6 pounds of cyanide which, at 23 cents per pound had a value of \$1,644. Adding to this the loss in dissolved gold amounting to \$2,672, the total loss is \$4,316, or 64 cents per ton.

These figures show clearly the weak points of the decantation system of slimes treatment. The Dakota Mill is one of the most successful mills in the district, treating what is practically the lowest grade of siliceous ores handled in the district. The results on sands at this mill are discussed under sand treatment. The treatment of the slimes at the Maitland Mill is similar to that at the Dakota. Each ton of dry slimes receives a treatment by 5.38 tons of barren solution and 0.96 tons of wash water. The solution going out with the slimes as moisture contain \$0.46 in gold per ton. The head slimes solution or the solution first decanted from the slimes while filling has a value of about \$2.00 per ton.

At the Lundborg, Dorr & Wilson mill the Moore slimes process is used on the slimes. There are three rectangular vats 15 feet long, 7 feet wide and 5.5 feet deep. The first tank has a double hopper bottom, the sides inclined at 45 degrees to more readily collect the heavy slimes which sometimes fail to be taken on the filter frames. There is a set of 35 frames 4.5 by 6 feet in area and made of two inch material. The filtering medium is 18 ounce duck. Both sides of the frames are effective as filters and the total filtering area is 1836 square feet. The interior of the frames are connected with a pump which produces suction and also with a compressor. The suction is equivalent to 18 inches of mercury. The set of frames is suspended from a hydraulic crane which transfers the frames from one vat to the other. The method of treatment is as follows: The slimes after agitation by air and a centrifugal pump in an eight foot sheet iron cone are run to the first tank of the Moore process, the frames are immersed in the slimes and the suction is started. A coating of slimes deposits on the filters and the clear solution is discharged by the pump. When the slimes layer on the filters has accumulated to the thickness of an inch, which amounts to a load of four tons on the set of frames, and takes from 40 to 65 minutes, the frames are lifted out with their adherent load of slimes. suction meanwhile being continued, and immersed in the next vat which is filled with barren solution. This barren solution is sucked through the slimes for 40 minutes, when the frames are transferred to the next vat which is filled with water. This water is sucked through the slimes for 40 minutes, when the frames are lifted out, transferred to above the discharge hopper, the suction changed to pressure, which causes the slimes to peel off into cars below the hopper. Some little scraping has to be done to clean the frames. No figures are as yet available concerning the results of the Moore process, permitting of a comparison with the decantation process. It is a fact however that the process discharges dryer slimes, those at the Lundborg, Dorr & Wilson mill containing from 34 to 36 per cent moisture, as against 50 per cent, which is the usual figure for the decantation process. At the mills where the upper layer of slimes is pumped off as described for the Hidden Fortune mill the slimes are discharged with 46 to 47 per cent moisture. Some of the mills of the district which must confine their tailings within narrow limits and cannot let them flow freely to waste, experience considerable trouble from the high moisture contents of their slimes tail-

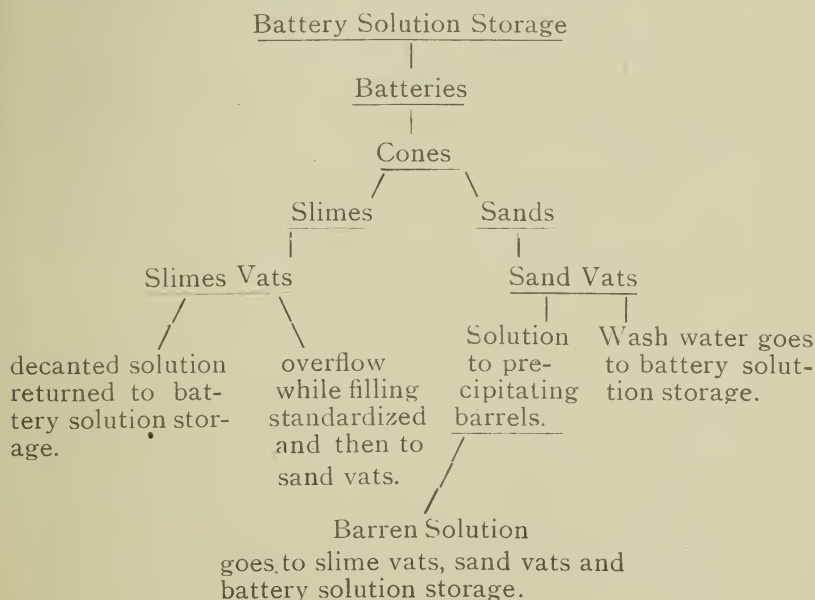


Filter Frames, Moore Slimes Process.

ings. The slimes tailings from the Moore process are much more easily held in check. In making a general comparison between the Moore and the decantation process it must be borne in mind that the slimes are under treatment in the Moore process only two to three hours, and in that time receive practically no agitation, so that the solution of the gold must take place practically before the slimes go to the Moore process. The Moore process, even with separate agitation however shortens the time on the slimes materially and a large capacity can be installed within a small space.

Filter press experiments have been made on a fair sized scale at one of the mills in the district, which indicated that slimes could be made containing about 25 per cent moisture and that these slimes, on account of the close washing feasible, carried very little cyanide and dissolved gold. It would not be surprising to eventually see filter pressing replace the decantation process at least in part.

THE DISTRIBUTION OF SOLUTIONS IN THE MILLS.—The distribution of solutions in the mills is quite complex, and the following diagrams show the general practice. Distribution of solutions at the Horseshoe mill.

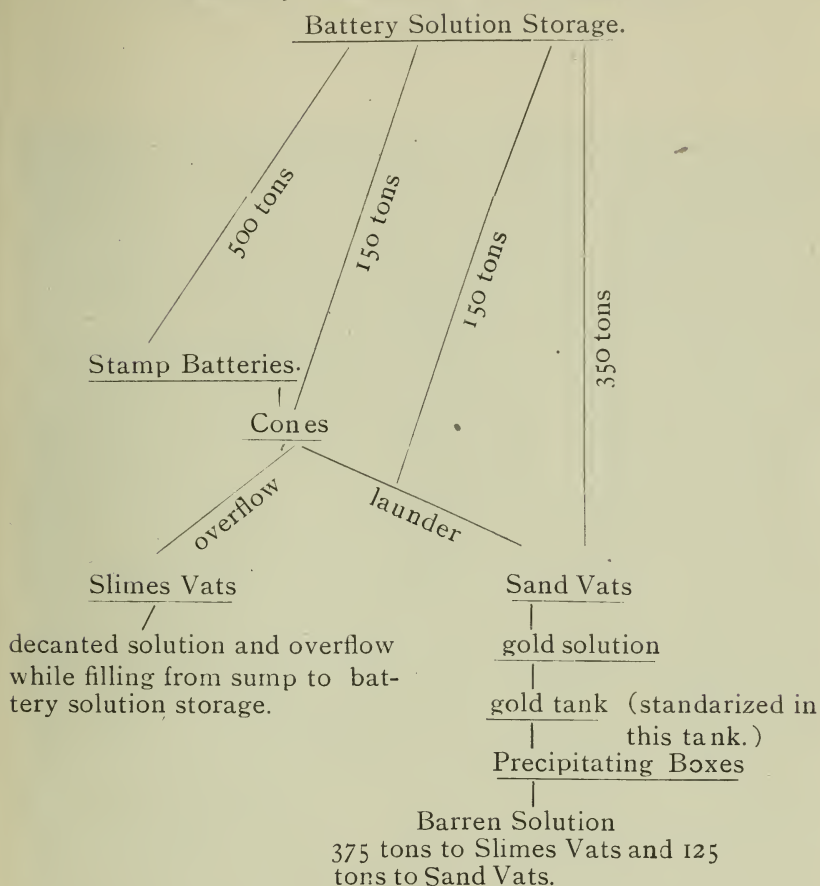


It will be noticed that the only solution going to the precipitating boxes is that which has passed the sands. This amounts

to from 300 to 1,000 tons per day. The battery solution has a gold value of approximately one dollar per ton. This value is derived for the greater part from the decanted slimes solution. The cyanide needed to bring up the strength of the solution is added to a comparatively small amount of decanted slimes solution, more particularly the overflow solution from the slimes vats while these are filling. This is brought up to three to four pounds of cyanide per ton and run on the sands so that these get the benefit of what is practically a strong solution.

In the scheme of the Maitland mill which follows, it will again be noticed that the only solution that is precipitated is that which has passed the sands. This is the general practice at all of the mills. The solution is restandardized ahead of the zinc boxes in order to get more efficient precipitation as there is some copper in the solution. The strength of the various solutions at the different mills is given under the discussion of the treatment of the sands. The battery solution in the case of the Maitland mill has a value of about 50 cents of gold per ton and the barren solution about ten cents per ton. About 1,100 tons of battery solution are pumped per day and close to 500 tons of solution are precipitated every day, which, after precipitation, becomes the barren solution, so that the mill handles per day about 1,600 tons of solution, which, with a capacity of 120 tons of ore per day, is 13.1 tons of solution per ton of ore. The pumping expense of a wet crushing mill per day is appreciable and in the design of a mill the question of handling the solutions is a very important one. At one of the mills the restandardization takes place in the battery sump so that the strongest solution is used in the battery. This practice is for obvious reasons not the best.

Distribution of Solutions at the Maitland Mill.

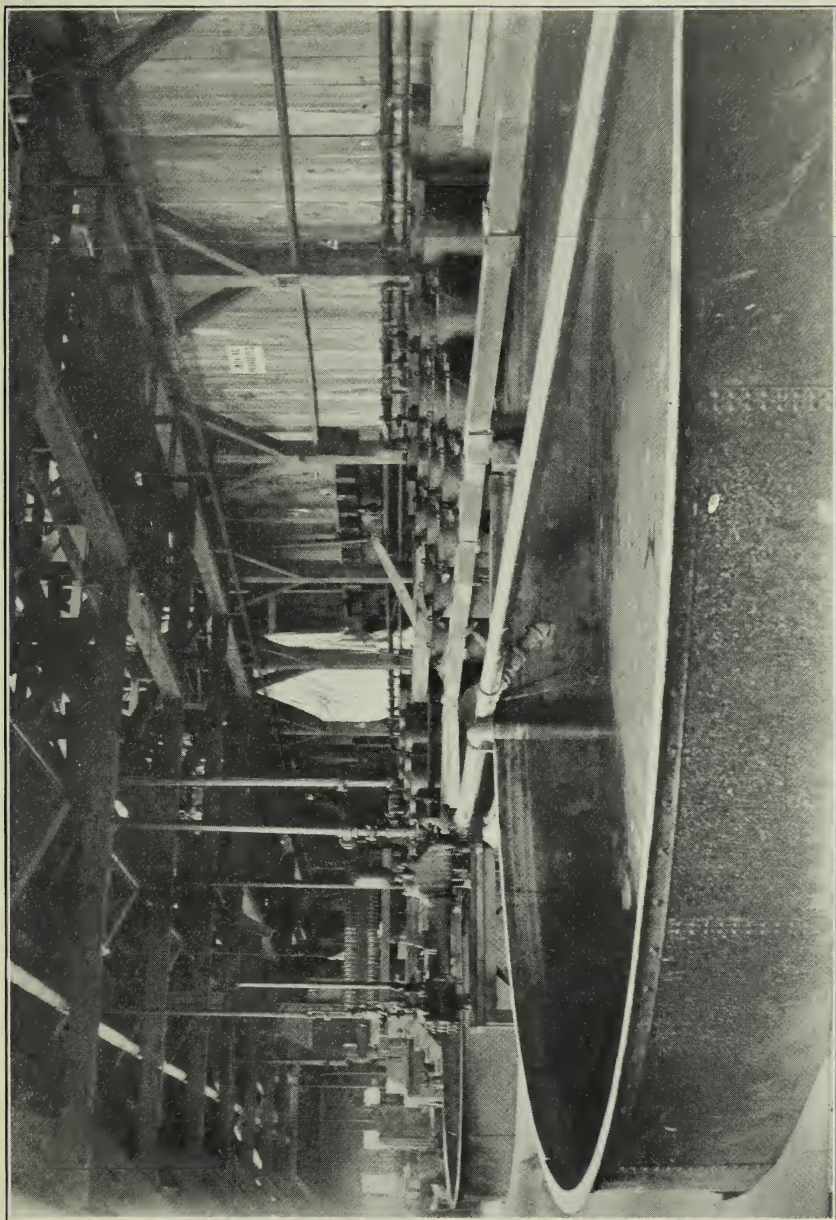


THE PRECIPITATION OF THE VALUES.— In general no troubles in precipitation are encountered in the form the process is employed at the present time. In the earlier practice of the mills when decanted slimes solution was passed through the zinc boxes with the idea of keeping the battery solution practically free from gold values trouble was experienced by getting a very bulky and low grade precipitate on account of the accumulation of fine ore slimes in the boxes, but in the present practice this trouble is avoided as described. At the Dakota and Horseshoe mills precipitation is carried on in barrels. These at the Horseshoe mill are two feet in diameter and two feet high, holding five cubic feet of zinc, or 25 pounds. 120 of those barrels are in use. The barrels can readily be taken up on

a pulley gear running on a trolley and conveyed to the clean-up tank for discharge and cleaning. At the Horseshoe mill where 225 to 250 tons of ore are treated per day, 1,000 tons of solution are precipitated every 24 hours. The zinc consumption is one pound per ton of ore treated. At the Dakota Mill it is 0.58 pounds. At the Hidden Fortune and the Maitland mills precipitation is carried on in the usual form of iron zinc boxes. At the Maitland Mill treating 120 tons per day there are four boxes of eight compartments each, the compartment having a capacity of seven cubic feet of zinc. From 450 to 500 tons of solution are precipitated per day, there being 2.13 tons of solution for each cubic foot of zinc. The solution entering the boxes is kept at 2.5 pounds of cyanide per ton, for a lower tenor causes trouble by copper precipitating, which the solutions carry to a small extent. The zinc consumption is 0.3 of a pound per ton of solution precipitated and 1.33 pounds per ton of ore treated.

TREATMENT OF THE PRECIPITATES.—The precipitates are refined at all of the mills by the usual sulphuric acid method into bullion which is disposed of to the United States Assay Office at Deadwood. The fineness of bullion varies at the different mills as the silver contents of the ores vary considerably. However, at most of the mills the bullion produced ranges between 450 and 600 fine in gold. In order to dispose of the bullion to the Government Assay Office at Deadwood, the bullion must have a minimum fineness of 600 in gold and at some of the mills in the smelting of the precipitates a matte is purposely formed so that the resulting bullion will be above the minimum limit. The slags from the smelting of the precipitates are disposed of to the smelters at Denver, although at the Horseshoe Mill the slags and matte from the smeltings are refined by melting with litharge and cupelling the resultant bullion.

GENERAL EXTRACTION FIGURES.—The extraction at the mills varies from 68 to 75 per cent, according to the ores treated. At the Maitland Mill where the extraction is 73.2 per cent and where approximately half the ore is treated as slimes and half as sands the extraction is distributed as follows, 39 per cent is extracted in the batteries and cones, 16.5 per cent in the slime treatment, and 17 per cent in the sand treatment. As a general thing the recovery of bullion is either two to three per cent higher or the same amount lower than the assays of the ores, tailings and solutions call for.



Precipitating Floor, Mogul Mill, Horseshoe Mining Company

THE COST OF TREATMENT.

The following is the detailed cost of treatment at the Dakota Mill during a period in 1902. The average value of the ore was \$4.70 per ton.

Labor.....	45.30
Superintendence.....	9.0
Cyanide.....	21.1
Zinc.....	3.3
Lime.....	1.2
Power.....	22.6
Shoes, dies, etc.....	9.5
Repairs.....	2.7
Refining.....	3.
Assay Office.....	4.
General Expense.....	5.0

Total..... 127.7 \$1.27 per ton.

The mill during this period treated 100 tons per day. The cost at the Dakota Mill, at the present time owing to a somewhat increased capacity has been reduced to \$1.17 per ton.

The cost at the Maitland Mill treating about 125 per ton per day, and which is not situated on the railroad so that the cost of supplies is considerably increased, is \$1.79 per ton. The cost at the other mills ranges between the costs at the two mills given.

NOTE. This paper is printed in advance, by permission, from the T. A. I. M. E., September 1904.

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